TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

Vol. 102

1932

GENERAL VOLUME

THIS VOLUME CONTAINS PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS HELD AT JOPLIN, OCTOBER, 1931 AND NEW YORK, FEBRUARY, 1932; OFFICERS, COMMITTEES, NECROLOGY, ABSTRACTS, LIST OF 1932 TECHNICAL PUBLICATIONS AND CONSOLIDATED INDEX

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Resumption of Serial Numbers for the Transactions

WITH Volume 99 of the TRANSACTIONS was resumed the serial numbering that was discontinued after the issuance of Volume 76 in 1928. The designation of the present volume therefore is TRANSACTIONS A.I.M.E., Vol. 102. The following table shows the existing titles of the volumes published in the interval and also the supplemental designations. Anyone desiring to revise the designations on the backbones of his volumes can obtain appropriate stickers with gilt letters, to match the binding, at small cost by addressing the office of the Secretary.

PRESENT TITLE OF VOLUME	ADDITIONAL DESIGNATION
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PREFACE

This volume of TRANSACTIONS is the sixth and final to be issued by the Institute during 1932. The six are as follows:

TRANSACTIONS, Vol. 97, Geophysical Prospecting.

TRANSACTIONS, Vol. 98, Petroleum Development and Technology.

TRANSACTIONS, Vol. 99, Institute of Metals Division.

TRANSACTIONS, Vol. 100, Iron and Steel Division.

TRANSACTIONS, Vol. 101, Coal Division.

TRANSACTIONS, Vol. 102, Metal Mining, Nonferrous Metallurgy, Nonmetallic Minerals, Mining Geology.

Residual TRANSACTIONS material on hand is principally a group of papers on milling and concentration that will be published in a special volume early in 1934. Five papers on Copper Metallurgy are to be included in a special volume of TRANSACTIONS for 1933 to be financed by the Rocky Mountain Income. The title probably will be the Rocky Mountain Fund Volume on Copper Metallurgy. Both pyrometallurgy and hydrometallurgy will be included, and a notable book on a special subject of great importance should result.

In the present volume, as a matter of permanent record, are published the roster of officers and the principal standing committees for the year; brief summaries of the proceedings of the Annual Meeting in New York and of the Division and Regional meetings held during 1932; and the necrology for 1931. Included is a complete index of material published by the Institute during 1932 in TRANSACTIONS, TECHNICAL PUBLICATIONS, PREPRINTS, and MINING AND METALLURGY, the monthly magazine, and also are to be found abstracts of papers published this year, except those appearing in this volume, and a classified list of TECHNICAL PUBLICATIONS and PREPRINTS.

After this volume the practice of publishing annually a so-called "General" volume of TRANSACTIONS is to be discontinued. The purpose is to avoid a definite waste in the cost of production and distribution. As can be seen from the list of heads under which come all of the technical papers published in this volume (for example) only about half the members of the Institute have a primary interest in it, except for the indexes and other general matter outlined in the preceding paragraph. Obviously a substantial waste is involved in the cost of printing and distributing a volume of this character to the entire membership. As an alternative the following plan has been adopted:

1. To publish annually a YEAR BOOK of about 150 pages, to contain the indexes and general matter, for distribution to every member. This

is to be bound with a flexible paper cover, but will be supplied with a regular stiff binding at nominal cost to any member so desiring.

2. To publish at five-year intervals a cumulative index of all publications, the book to have the regular stiff binding.

3. To publish TRANSACTIONS volumes at intervals when material is available to cover one or more of the "general" subjects; as, for instance, a volume on Nonferrous Metallurgy or one jointly on Metal Mining and Milling and Concentration. These volumes will be chosen only by members directly interested. The result will be greater discrimination and selectivity in distribution with a consequent more effective expenditure of publication funds.

The foregoing plan was approved by the Directors after being submitted to them in the form of a recommendation by the Papers and Publications Committee. In order to ascertain the opinion of the membership at large, the proposal was submitted by the Papers and Publications Committee to 45 members residing in various parts of the country. The list included the chairmen of Divisions, the chairman and either the vice-chairman or secretary of each Technical Committee, and one member picked at random from each committee. The response was almost unanimous in favor of the plan and the Committee was unanimous in making the recommendation. It is believed that the new program is in the best interests of the greatest number and that the adoption of it will meet with general approval.

A. B. PARSONS, *Secretary.*

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New York Meeting

The 141st meeting* of the American Institute of Mining and Metallurgical Engineers was held in New York, Feb. 15 to 18, 1932. It consisted of the annual business session, 35 technical sessions at which 167 papers were presented, two round table discussions, two sessions of the Board of Directors, three meetings of Section delegates, three formal lectures, six group dinners and nine group luncheons, luncheon daily for members and guests, thirteen committee meetings and conferences, the annual reception and dinner dance, an informal dance, a smoker-dinner and two meetings of the Woman's Auxiliary. The portrait of Herbert Hoover, President of the United States, painted by Philip A. de Laszlo at the request of the four Founder engineering societies, was unveiled at a special meeting on Monday afternoon.

The Institute of Metals Division held four general sessions, a dinner at the Commodore Hotel, at which Dr. J. A. Gann was the principal speaker, and its annual lecture, which was delivered by Dr. Paul D. Merica. The lecture was entitled, "The Age-hardening of Metals." The Executive Committee met at a luncheon on Tuesday.

The Iron and Steel Division held three general sessions, a conference on Blast Furnace and Raw Materials and a Division luncheon. E. C. Bain delivered the Howe Memorial Lecture, the title of which was "On the Rates of Reactions in Solid Steel." The Executive Committee met Thursday afternoon.

The Institute of Metals and Iron and Steel Divisions joined in holding two sessions of a Gases in Metals Symposium. These Divisions also met jointly with the New York Metropolitan District members of the American Society for Testing Materials at a session on Thursday evening.

The Coal Division held two sessions on Coal Classification and a round table on Coal Lands Valuation. C. B. Huntress, Executive Secretary, National Coal Association, addressed the Division at its luncheon on Monday.

The Petroleum Division held six sessions, including a joint meeting of the Engineering Research and Production Engineering sections, the usual Production Symposium and a session on Petroleum Stabilization. The Division dinner was held at the Commodore Hotel Thursday evening and was followed in the same room by the usual annual production and technology review session. Amos L. Beaty, president of the American Petroleum Institute, talked informally and four short papers were presented.

The Committee on Geophysical Methods of Prospecting required three sessions for its extensive program of papers. The Committee on Mining Geology held two sessions, one a symposium on Ferroalloy Materials. The Committee on Milling Methods also held two sessions, the second a

* For preliminary story of meeting see MINING AND METALLURGY (February, 1932) 67; for news story see the number for March, 1932.

round table discussion on Crushing and Grinding. A notable program was that of the Committee on Nonmetallic Minerals, a morning session with papers on the Beneficiation of Nonmetallics, and an afternoon session with papers of more general interest.

Other committees held sessions as follows: Mining Methods, two; Mine Ventilation, one, and a meeting of the Committee on Mine Ventilation Code; and Nonferrous Metallurgy, two, covering the Reduction and Refining of Copper, Lead and Zinc. The Committee on Rare Minerals and Metals sponsored a formal lecture which was delivered by Dr. Fred Allison. It was on "The Magneto-optic Method of Analysis with Particular Reference to the Detection of Elements 85 (Alabamine) and 87 (Virginium) and the Heavy Isotope of Hydrogen." At a session of the same committee on Tuesday afternoon, Hugh S. Spence talked to a large audience on "The Great Bear Lake Uranium Deposits." On Tuesday afternoon the Committee on Engineering Education organized as the Mineral Industry Education Division of the Institute. The Committee on Correlation of Research met at luncheon on Thursday.

At the annual business meeting on Tuesday afternoon, reports of the President, Treasurer and Secretary and of the respective Committees on Admissions, Membership, Papers and Publications, and Library were presented. The report of the tellers showed the following to be elected as officers for the year 1932: Scott Turner, President; Frederick M. Becket, Vice-president; Paul D. Merica, Vice-president; Erle V. Daveler, Eugene McAuliffe, H. S. Mudd, J. B. Umpleby and Charles C. Whittier, Directors. At an executive session of the Directors on Tuesday afternoon, Edgar Rickard was chosen to fill the unexpired term of Mr. Turner as Vice-president and Director; Karl Eilers was re-elected Treasurer, and A. B. Parsons was re-elected Secretary.

Twenty-two Sections and three Divisions sent delegates to the Annual Meeting. They held three sessions and were guests at the Directors' dinner on Tuesday evening and at the Directors' meeting afterward.

The sixteenth annual meeting of the Woman's Auxiliary was held Tuesday morning and afternoon with 99 registered from the metropolitan area and 78 from other sections throughout the United States. The work of the Educational Scholarship Committee was indicated in various reports: twenty-six men have been graduated, seven men were then receiving assistance, and three new scholarships had been awarded for the year 1932–33. The Library Committee reported that 5839 books had been distributed throughout the country during the year.

A dinner-smoker was held Monday evening in the ballroom of the Commodore, at which 569 were present, and an informal dance, also well attended, was held at the Commodore on Tuesday evening.

More than six hundred members, friends and ladies attended the annual dinner and dance, Wednesday evening at the Hotel Commodore.

The incoming and retiring presidents held a reception preceding the dinner. G. Temple Bridgman was toastmaster. The roll of the Class of 1882 of the Institute Legion of Honor was called. The insignia and diploma of the Legion were presented to E. L. Herndon and William H. Hulick; the others were not present. The James Douglas medal was presented to Champion H. Mathewson and the William Lawrence Saunders medal to F. W. Bradley. The Robert Woolston Hunt prize was awarded to Howard Scott. Ora E. Clark was the recipient of the J. E. Johnson, Jr., Award. The Alfred Noble Memorial Prize was presented to Corbin T. Eddy, the first recipient of this prize. H. Foster Bain, chairman of the Charles F. Rand Foundation Committee, announced the completion of the Charles F. Rand Foundation fund of $10,000 to be held and administered by the Institute. Retiring President Robert E. Tally made his farewell address. Scott Turner, the incoming President, spoke briefly.

Los Angeles Meeting

A Western Regional Meeting* of the Institute was held at Los Angeles, July 28 and 29. The first day was devoted to Petroleum, the second day to Gold and Silver. The petroleum papers were arranged for by the Petroleum Division through the Southern California Section; papers for the sessions on Gold and Silver were provided by the San Francisco Section. The Biltmore Hotel was headquarters. More than two hundred and fifty registered.

The Thursday morning proceedings consisted of a discussion of Basic Economic Principles Applied to the Petroleum Industry and a symposium on Recent Work on Porosity and Related Factors. In the afternoon there was a symposium on Present and Proposed Bases of Proration. On Friday morning there was a Gold session with four papers; in the afternoon a Silver session with two papers.

A dinner at the University Club on Thursday evening was attended by President Turner, Directors Krumb, Mudd, Roberts and Tally, Secretary Parsons and thirty others, the latter mostly members of the two California Sections. The informal dinner at the University Club on Friday evening was attended by more than one hundred men and women. The button and diploma of the Legion of Honor were presented to William Lawrence Austin and Walter W. Wishon.

During the meeting, the ladies of the Auxiliary were taken to the Hollywood Bowl to enjoy the "Symphonies under the Stars," to dinner at the Wilshire Country Club, and to the Huntington Library. On the days following the meeting, all could see the opening ceremonies and events of the Olympic Games, which started on July 30 and continued to August 14.

* For news story see MINING AND METALLURGY (September, 1932) 406.

Ponca City Meeting

Besides participating in the program of the Los Angeles Meeting,* the Petroleum Division held its regular fall meeting* at Ponca City, Sept. 30 to Oct. 1. Approximately two hundred attended. The Conoco Club was headquarters. All of Friday, from 9.45 a. m. to 10 p. m., with time out for luncheon and dinner, was devoted to papers and discussion on Rationalization. At dinner in the evening, Vice President Curtis was the guest of honor. The morning session on Saturday included a discussion of various proration methods and papers on production technique. In the afternoon there was a symposium on bottom-hole pressures.

Buffalo Meeting

The Institute of Metals and Iron and Steel Divisions met† jointly at the Hotel Statler, Buffalo, N. Y., Oct. 4 and 5, during the week of the National Metal Congress (Oct. 3 to 7). Approximately two hundred registered. The Institute of Metals Division held two technical sessions and the Iron and Steel Division one. The two Divisions held a joint session on Theoretical Metallurgy and also met for the annual Science Lecture which was delivered by Dr. A. W. Hull on the subject of "New Vacuum Valves and Their Applications." The joint dinner of the Divisions was held at the Hotel Statler on Wednesday evening. Dr. Zay Jeffries gave a nontechnical talk on "Tungsten." The Executive Committees of the Divisions met separately at luncheon on Thursday.

Hazleton Meeting

The Coal Division met‡ jointly with the Pennsylvania Anthracite Section of the Institute at Hazleton, Pa., Oct., 14 and 15, with the Hotel Altamont as headquarters. Two hundred and seven members and guests registered. There were two technical sessions on Friday and a dinner in the evening which was attended by more than two hundred. Charles W. Wright, of the Bureau of Mines, presented the Sentinels of Safety trophy to the Jeddo-Highland Coal Co., for its safety record. Most of those attending the meeting, accompanied by many of the ladies, visited the No. 8 (Coaldale) colliery of the Lehigh Navigation Coal Co. Some of the members also visited the Jeddo No. 7 Chance preparation plant of the Jeddo-Highland Coal Co. Opportunity was also offered to see a gas producer plant and anthracite stripping operations. The entertainment for the ladies included an automobile ride about the beautiful country surrounding Hazleton, and a tea at the residence of Mrs. Alvin Markle.

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* For news story of Los Angeles meeting see MINING AND METALLURGY (September, 1932) 406; for news story of Ponca City meeting see MINING AND METALLURGY (November, 1932) 484.

† For news story of meeting see MINING AND METALLURGY (November, 1932) 478.

‡ For news story see MINING AND METALLURGY (November, 1932) 492.

Necrology

The following is a list of members who died in 1931. It is compiled from reports to the Secretary's office. Biographical sketches published in Mining and Metallurgy are indicated in the last two columns.

Year of Election	Name	Date of Death	Issue Containing Biography 1931*	Page
1914	Agnew, John L	July 8	August	382
1899	Ahlers, Rudolph O	April 16	June	295
1920	Allen, Andrews	March 21	May	252
1930	Andrus, Dexter E	Aug. 6	February*	100
1916	Badgley, Charles W	July 15	September	423
1902	Barber, George M	Jan. 12	March	171
1907	Bell, Hugh	June 26	August	381
1898	Beyer, Samuel W	June 2	July	338
1907	Blackmer, William D	Aug. 5	September	422
1914	Brennen, William D	Nov. 1	December	546
1916	Brewster, Thomas T	Dec. 13	January*	54
1928	Burr, Samuel P	April	June	294
1917	Burrage, Albert C	June 28	August	384
1906	Carlton, Albert E	Sept. 7	November	506
1925	Carr, Herbert J	March 13	August	381
1917	Carroll, Alexander W	Dec. 17	February*	100
1917	Chapman, George Albert	Aug. 15	November	508
1927	Chilson, Ernest	Jan. 4	February	115
1902	Clement, Harrison E	June 24	August	382
1926	Comstock, William O	Sept. 8	November	508
1913	Corwin, Frank R	Sept. 15	November	507
1921	Cosh, A. D	January	March	170
1928	Crampton, Theodore H. M	Sept. 30	November	506
1907	Cunnyngham, Floyd E	May	February*	100
1917	Davis, Thomas	Jan. 24	March	171
1929	Dawson, Thomas W	Dec. 19	February*	99
1896	Dean, A. L	Dec. 23	February*	99
1892	DeKalb, Courtenay	Sept. 2	October	462
1889	Devereaux, James H	June 13	August	382
1900	deWendel, Charles	March 3	April	211
1919	Dresser, Carl K	Feb. 1	April	210
1889	Edison, Thomas A	Oct. 18	November	508
1906	Edwards, Victor E	May 16	June	295
1929	Eliot, Walter G	May 3	June	295
1918	Emerson, Harrington	May 23	July	338
1924	Emory, Lloyd T	January	March	171
1921	Eubanks, George	Dec. 5	January*	54
1926	Farnham, Sidney W	March 12	May	252
1927	Forstner, van D., William M	June 15	October	462
1918	Franklin, Nelson	June 14	August	381
1930	Galbreath, Neil M	April 11	June	295
1895	Granger, Henry G	Aug. 8		
1922	Greensfelder, Nelson S	April 5	May	252
1902	Hall, Everett Joel	Sept. 2	October	461
1893	Hamilton, Frank C	January	December	546
1881	Hart, Edward	June 6	August	382
1917	Harvey, Alexander S	Oct. 16	January*	54
1927	Harvey, Roger D	August		
1913	Hess, Rush M	Aug. 14	September	422
1922	Holden, Stephen M	Aug. 22	November	507
1885	Humphrey, Charles	June 18	September	422
1920	Huneke, Albert J	June 27	August	382
1894	James, Alfred	Sept. 5	December	546
1916	Jenkins, Charles V	March 17	June	294

* An asterisk indicates that the biography appeared in 1932.

Year of Election	Name	Date of Death	Issue Containing Biography 1931*	Page
1888	Kedzie, George E	Oct. 8	December	545
1922	Kilinski, Edward A	August	October	461
1915	Kirchen, John G	March 4	May	252
1909	Kitson, Howard W	Aug. 7	October	461
1888	Larue, William G	June 9	January*	54
1920	Longley, A. L	Feb. 13	April	210
1923	Longyear, Philip O	April 22	June	295
1919	Mackenzie, George C	Aug. 22	October	461
1926	Mancha, Raymond	Jan. 3	February	115
1907	Mandell, Ambrose J	June 28	February*	100
1885	Markle, Alvin	March 19	June	294
1879	Mather, Samuel	Oct. 18	November	508
1918	Moore, Richard B	Jan. 20	February	115
1906	Morse, Bradish P	Dec. 28	February*	99
1900	Morton, Roscoe B	July	September	422
1921	Newman, Lewis D	March		
1918	Parr, Samuel W	May 16	November	507
1928	Pettengill,† H. J	Jan. 12		
1889	Penrose, R. A. F., Jr	July 31	September	423
1921	Poston, Elias M	Oct. 9	December	545
1929	Ritter, John P	Sept. 22	February*	99
1886	Robertson, James D	Aug. 25	October	462
1919	Rogers, Robert B	June 7	February*	100
1914	Rusterholz, Rudolph W	June 17	November	508
1928	Ryan,† Richard S	Nov. 28	November*	501
1916	Sanders, W. Murray	April	June	295
1906	Saunders, William L	June 25	August	383
1898	Skinner, Orville C	May 19	June	295
1920	Soderberg, Olaf A		March	170
1914	Solomon, Frederick W	March 8	June	294
1911	Stevenson, George E	Jan. 3	April	211
1927	Sveshnikoff, Alexander P	Aug. 7		
1904	Thomas, Kirby	June 22	August	382
1924	Timmis, Frank W	Feb. 6	April	211
1924	Tower, Joseph T., Jr	Aug. 23	November	507
1915	Walter, Raymond A	Nov. 3	December	546
1907	Wayland, Russell G	Dec.	January*	54
1903	Wells, Bulkeley	May 26	July	339
1908	Wells, James S. C	Sept. 29	December	545
1896	Wiley, Walter H	May 16	June	294
1915	Williams, William H	Oct. 14	November	507
1913	Woodhouse, Charles C., Jr	Nov. 7	December	546
1925	Worden, Beverly L	April	November	507
1897	Wright, Lewis	April 29		
1920	Yonekra, Kiyotsugu	Oct. 26	April*	190
1912	Young, Hayes W	March 26	May	252

Since the beginning of January, 1931, the following deaths also have been reported to the Secretary's office:

Year of Election	Name	Date of Death
1887	Dinkey, Charles E	Sept. 8, 1928
1882	Dumont, John M.	January, 1930
1928	Hall,† W. H.	1929
1928	Hodgens, Thomas M.	1928
1920	Perry, F. E.	Nov. 13, 1929
1928	Reed,† W. B.	June 20, 1930

* An asterisk indicates that the biography appeared in 1932.
† A Rocky Mountain Club member of the Institute.

New Vacuum Valves and Their Applications*

By A. W. Hull,† Schenectady, N. Y.

(Buffalo Meeting, October, 1932)

The new valves described in this article are the latest product of the Research Laboratories of the General Electric Co. Some of them are still in the laboratory stage, others have already found important applications. Some of the larger ones, which have graduated from the laboratory, are waiting for their apprenticeship in industry, there to refine their frailties and complete their preparation as electrical servants. The career that awaits them, though dimly foreseen, appears fascinating. And as surely as history repeats itself, scenes still more romantic, now beyond the horizon, will unfold as we approach them. This is the romance of science.

Counting Electrons

Let the smallest tube lead the march of review. It is a special Pliotron, designed to measure currents smaller than any yet detected. It goes by the unromantic name of Pliotron FP-54. Fig. 1 shows this miniature tube—miniature in performance rather than size, for it is the size of an ordinary radio receiving tube. Its appearance has nothing distinctive, except the quartz beads above and below the plate, which support the grid. They minimize insulation leakage.

In principle, the FP-54 is like any *grid-controlled* high-vacuum electron tube; but in structure and operating characteristics it is entirely special. An extra "space-charge" grid, maintained at 3 volts positive with respect to the filament, holds back the small but pernicious current of positive ions emitted by the filament; the normal plate voltage is 6 volts, grid bias 3 volts, thoriated-tungsten filament temperature 1700° K., plate current 40 microamperes. These conditions are essential to the avoidance of grid-currents, which may consist not only of insulation leakage and positive ions from the hot filament and from residual gases, but of high-speed electrons from the filament, photo emission from the grid due to the light of the filament, and electron emission caused by X-rays generated by the impact of electrons on the plate.

* Science Lecture delivered before the Institute of Metals Division and the Iron and Steel Division of the A.I.M.E.

† Assistant Director, Research Laboratory, General Electric Co.

The present sensitivity of this new tube is slightly better than that of the most sensitive electrometer, over which it has an enormous advantage in being less fragile. In amperes this sensitivity, that is, the smallest current that can be measured, is $\frac{1}{1,000,000,000,000,000,000}$. The ampere is evidently an inconveniently large unit for our purpose. In terms of the smallest known unit, the electron, the sensitivity is approximately *six electrons per second.*

This microtube has the distinction of being entirely impractical. Its applications, present and future, as far as eye can see, are purely scientific. It counts cosmic rays. It measures, in cooperation with the photoelectric cell, the light from distant stars, being able at present to detect the light from a star of the fourteenth magnitude. It records the fragments—neutrons, protons, and alpha particles—of atomic nuclei smashed by high-speed ions. The structure of these atomic nuclei, the 92 hitherto indivisible elements of atoms, appears to be the next objective of scientific research, the next nature-fortress which science aspires to storm. Perhaps our diminutive tube may be the sling with which some scientific David shall make this conquest.

Measuring Nerve Messages

Next in review comes another impractical tube, likewise diminutive in function and ordinary in appearance; known as PJ-11 (Fig. 2). It measures voltages 10 times smaller than could be detected before. It, too, is a device of most ordinary structure and form; a simple three-element tube of standard size and construction. Its special feature is good vacuum—naturally invisible.

Good vacuum is a relative term. The vacuum of 1880, used by Hittorf and Crookes, was about $\frac{1}{1000}$ atmospheric pressure. It was sufficient to reduce the sparking potential between electrodes 1 cm. apart from 30,000 volts to about 300 volts. At this pressure electrons have long free paths and easily attain speeds at which they *ionize* the atoms they strike, and ions and electrons play about equal roles in carrying the "glow-discharge" current. Such glow-discharges have found many applications—for rectification, relays, illumination, television.

The "good vacuum" of 1900 was 1000 times better; that is, about $\frac{1}{1,000,000}$ atmospheric pressure. Electrons in such a vacuum only occasionally meet atoms, and the ions formed by these encounters contribute only a fraction of a per cent to the current. The current through the vacuum might be called a *pure electron current,* since at least 99.44 per cent of it consisted of electrons. This vacuum was good enough to enable Lenard and others to solve the mysteries of photoelectric emission, J. J. Thomson to discover and identify the electron. and O. W. Richardson to

discover the laws of thermionic emission. It enabled Fleming to invent the Fleming valve, and DeForest the audion.

Langmuir, in 1912, made a discovery. The so-called pure electron currents of Richardson and others had always been limited by the electron emission of the filaments, the only known limiting factor. Langmuir, having a new and better filament material, tungsten, to play with,

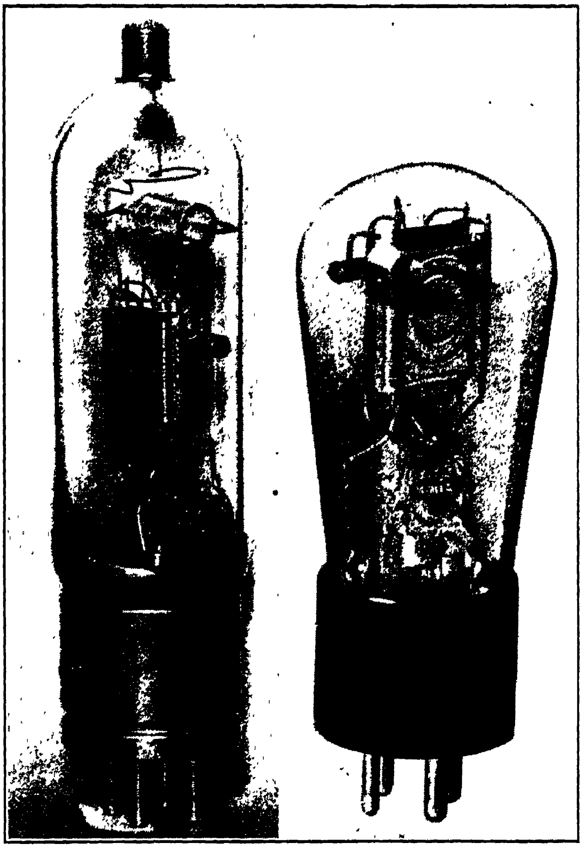


FIG. 1. FIG. 2.

FIG. 1.—PLIOTRON FP-54. THIS IS CAPABLE OF DETECTING CURRENTS AS SMALL AS 10^{-18} AMPERES.

FIG. 2.—PLIOTRON PJ-11, A *low-noise tube* DESIGNED FOR MEASURING SMALL VOLTAGES.

decided to test this limitation as far as possible. He tried a tungsten filament with an emission of 100 amp. and obtained a current of only $1/100$ amp. Why? He had discovered a *new limitation*, which he found to be caused by the mutual repulsion of the electrons in the vacuous space, and which he termed "space-charge limitation." Briefly stated, electrons repel each other, according to Coulomb's law, being charges of like sign. Hence they march across the vacuum in very open array, far enough apart

to be seen with the naked eye.[1] But, Langmuir discovered, this space-charge condition is present only in a *Langmuir-pure* electron discharge, 100 times purer than that of Richardson. For it is the electron-attracting *effect of the presence* of positive ions, not the current they carry, that is important for space charge. A vacuum so good as to give a negligible positive ion *current* (less than 0.25 per cent of the electron current) may have as many ions *present* as there are electrons, since the ions move several hundred times more slowly than electrons. In such a vacuum the space-charge limitation, due to electron repulsions, is entirely lacking. It is just this vacuum, and its freedom from space-charge limitations, that is utilized in the Thyratron tube.

The importance of this distinction lies in the fact that the whole foundation of amplifying action rests on space-charge. As long as tubes were merely rectifiers, like the Fleming valve, pure *current* was a sufficient specification of vacuum. With the advent of amplifiers, *pure space-charge* became the fundamental criterion.

All this is old history. The vacuum tube of today may say of Langmuir's vacuum specifications, "All these things have I done from my youth up. What lack I yet?" The answer is, still more purity. When we amplify enough—the sky is the limit with cascaded screen-grid tubes—we soon begin to hear the tube itself. A steady hiss, of no particular frequency, is what it sounds like in telephones. A milliameter instead of telephones tells the same story; the needle dances irregularly, wildly. Part of this "tube noise" we now understand and recognize as fundamental. It is due to the graininess of electricity. It is obviously impossible to have a smooth flow of anything that is made up of finite grains, like electrons. This so-called "shot effect," the irregularity of the patter of electrons on the plate, can be calculated and measured, electrically. It is equivalent, in an ordinary amplifying tube, to an input signal of about 1 microvolt. We must accept this as a fundamental limit to the smallness of signal that is worth amplifying, since anything smaller will be hissed down by the shot effect.

But there are other tube noises, which are particularly evident at frequencies below 1000 cycles, that are 100 to 1000 times larger than the shot effect. These, the worst of them, at any rate, have now been traced to bad vacuum, to ions produced in the still residual gas of ordinary amplifying tubes. The cure was more vacuum, another stage of vacuum perfection. The result is the PJ-11, with a low-frequency input noise level of less than ½ microvolt (practically all shot effect), as compared to about 10 times this value for the best tube previously available.

No one knows what will be done with this modest addition to the tube family. The possibility of amplifying and observing things 10 times

[1] The average distance apart of electrons in an ordinary receiving tube is about 1⁄100 mm., which is the diameter of the filament in a standard 15-watt Mazda lamp.

smaller than before suggests new discoveries; perhaps in the field of physiology, by measuring heart-beats, nerve-impulses, thought-waves.

Power Pliotron Tubes

Radio transmission has made available a fine line of power Pliotrons, from 5 watts up to 500 kilowatts. But the field has been too small for quantity production, with its simplifications and economies. There is now appearing on the horizon a need for power Pliotrons in industry. High-frequency heating of metals; high-frequency heating of people, at present for producing benign fevers that have been found effective in the treatment of paresis and arthritis, but eventually, perhaps, in lieu of coal; sterilization of milk, grain, fruit, bulbs; production of ozone; these are possible markets. Pliotrons suitable for these purposes can hardly be called accomplishments, since they are not yet available, but demand, if it calls, will find them ready.

Chemical Analysis

X-ray chemical analysis has many unique features. As a qualitative analysis it is infallible; that is, the presence or absence of the character-

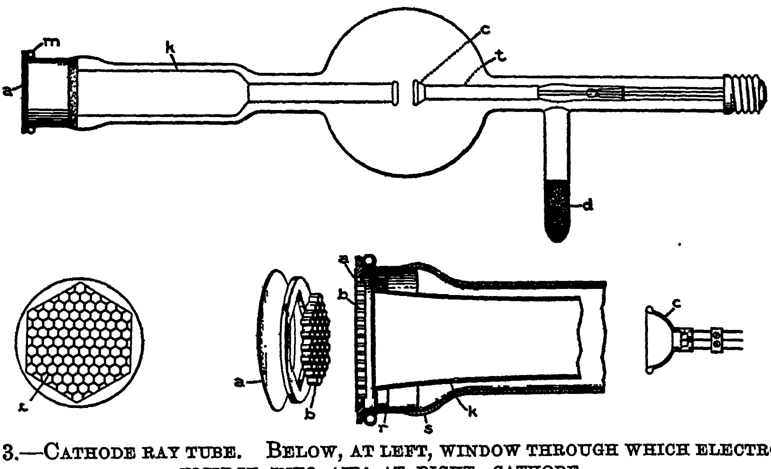


Fig. 3.—Cathode ray tube. Below, at left, window through which electrons emerge into air; at right, cathode.

istic lines of an element in the spectrum is absolute proof that the element is or is not present to the extent of a fraction of a per cent; it supplements chemical analysis, being especially applicable to elements of high atomic weight, from aluminum to uranium, which are least easily analyzed by other methods; for some of the rare elements it is the only reliable method; and the results are easily interpreted, since the number of lines in the X-ray spectrum is so small that they are readily identified.

But the technique of this analysis has been tedious. There has been no satisfactory way to obtain X-rays from the specimen except to put it inside the X-ray tube.

The problem has been solved by the new cathode ray tube (Fig. 3), which makes it possible to produce X-rays in air. A cathode, similar to that in an X-ray tube, focuses a beam of electrons on an aluminum window that closes the tube. Parts of the window are stamped very thin, so that the electrons pass through without appreciable resistance, into the air, and travel several inches in air before they are stopped. The substance to be analyzed is placed near the window, which is grounded (Fig. 4). Electrons emerging from the window strike the sample and excite its characteristic X-rays. These are resolved into a spectrum by a crystal, and the characteristic X-ray spectral lines are photographed, or are measured electrically by an FP-54 Pliotron.

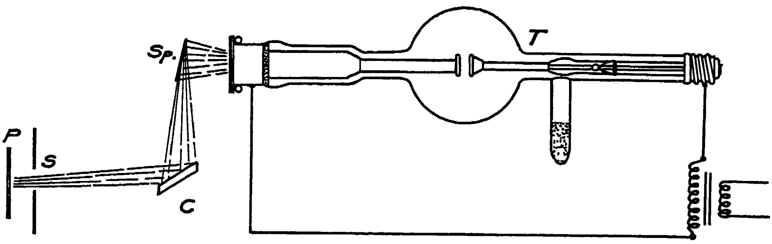


FIG. 4.—DIAGRAM OF APPARATUS FOR X-RAY CHEMICAL ANALYSIS, USING A CATHODE RAY TUBE TO EXCITE THE X-RAYS.

T, cathode ray tube; *Sp.*, specimen to be analyzed; *C*, calcite crystal; *S*, slit; *P*, photographic plate.

ELECTRIC EYES

The photoelectric cell is old—middle-aged, one might say, since it has passed its fortieth birthday. During the past five years, however, it has done most of its growing, stimulated by the small but important application in talking movies. The result is the caesium photo tube, known as PJ-23 (Fig. 5), 100 times more sensitive than previous cells, with the important property of being able to see red—for there is much more red light than blue in the world.

The special importance of this new electric eye is as a team mate for another new tube, the Thyratron. By itself, the PJ-23 is scarcely able to do any useful work. It gives a response of only 40 microamperes, at best, when illuminated with one lumen of electric light, which is about the amount it receives from a 50-watt lamp at a distance of 6 in. This current is too small to operate reliable relays, even slow ones. But it is ample for the grid of the Thyratron, its giant mate. Together they are a high-speed, high-power team, ready to contract for any type of electric service where reliable, quick action is required in response to light signals.

The Thyratron

The Thyratron tube is a grid-controlled arc discharge device. Like the Pliotron, it is a "three-element tube in which the flow of electrons between cathode and anode is controlled by voltage applied to a grid." But it differs from the Pliotron in two important respects. The first is "purity." The electron current in the Pliotron must be very pure; the

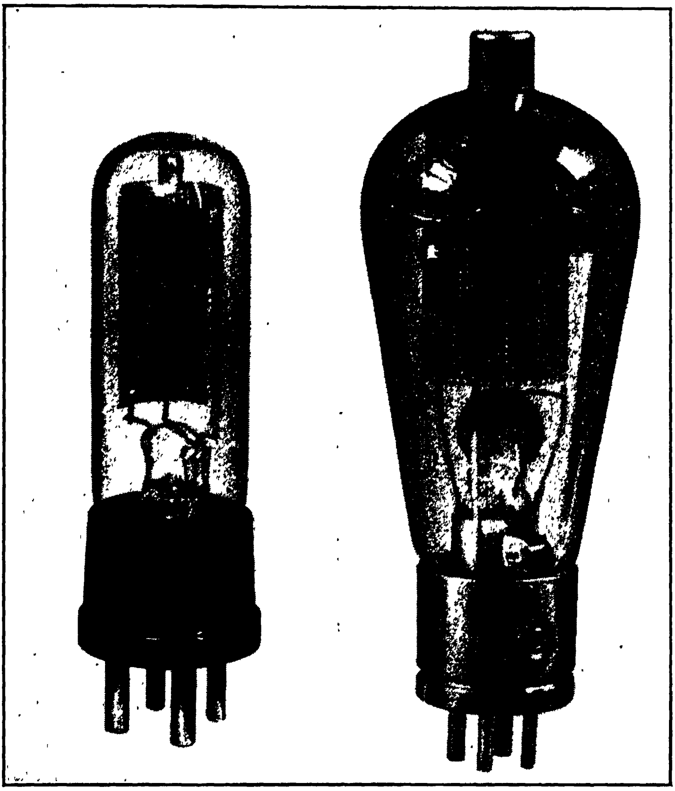


Fig. 5. Fig. 6.
Fig. 5.—Caesium phototube, PJ-23.
Fig. 6.—Thyratron FG-17, designed for currents up to ½ ampere at 2500 volts maximum.

presence of a few positive ions spoils the space-charge-limiting repulsions on which its whole action depends. The electron current in the Thyratron, on the other hand, must be 100 per cent impure; that is, it must have as many positive ions as electrons. Electron repulsions are so completely neutralized by the presence of the positive ions that 10,000 times as many electrons can be crowded into the space; there are between 10^{12} and 10^{13} electrons per cubic centimeter in the current stream of the Thyratron, compared with about 10^9 in the Pliotron. Hence the Thyratron can carry

larger currents with lower voltage drop—amperes instead of millamperes, with a few volts, from 10 to 18, between anode and cathode, instead of hundreds of volts.

The second fundamental difference between the Thyratron and Pliotron is that the grid of the Thyratron only partially controls the current flow. It can control only the *starting* of the current. After the current has started, the grid is nearly powerless; under ordinary conditions it neither controls the magnitude of the current, nor stops it. But if the current stops, the grid can prevent it from re-starting, or allow it to restart, at will. Inability to control the magnitude of the current is not a limitation, for this control must necessarily be sacrificed to obtain efficiency—the two are mutually exclusive. Inability to stop the current is a limitation, though not a serious one, since the *rectifying action* stops the current whenever the anode voltage reverses. Such reversal takes place periodically in many circuits—as, for example, when alternating voltage is used for the anode—and when it does not, can always be made to take place by starting another Thyratron.

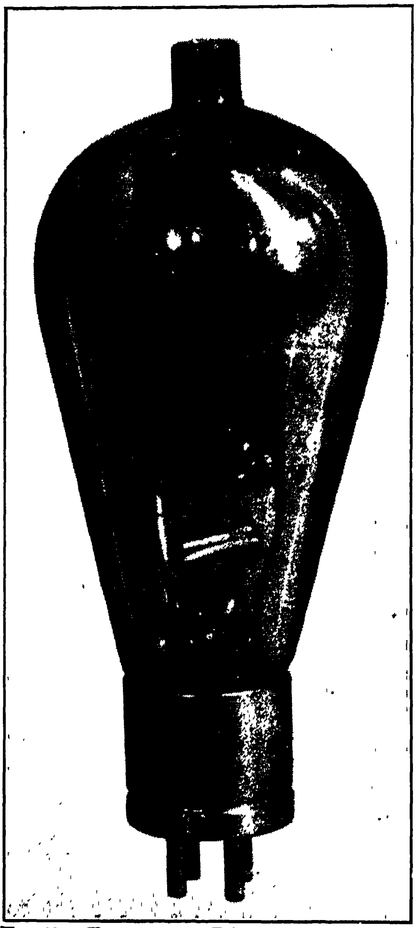
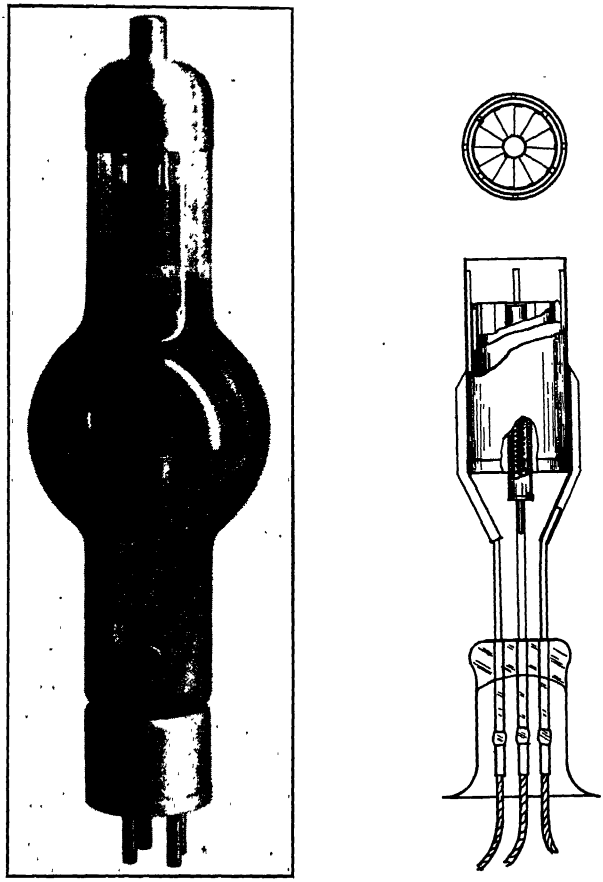


FIG. 7.—THYRATRON FG-57, RATED AT 2½ AMPERES AVERAGE, 1000 VOLTS, MAXIMUM.

The reason that a negative grid ordinarily cannot stop or control the discharge is that it attracts to its neighborhood enough positive ions to neutralize exactly its own negative charge, so that its negative influence extends only a short distance. It will always insulate itself in this way if there are positive ions in the space, but cannot if there are no positive ions. Hence, in order to regain control, the discharge must be stopped only long enough to allow the ions to diffuse to the walls. This time varies, according to grid mesh and vapor pressure, from about 50 to 200 microseconds.

Typical Thyratrons are shown in Figs. 6 to 10. The FG-17 (Fig. 6) is the simplest type. It has an oxide-coated filament, like the ordinary

Pliotron, but its grid is larger and farther removed from the filament. The reason for this is twofold. First, closeness is less necessary than in the Pliotron, because the current is not limited by electron space charge. Second, distance is desirable in order that the grid should remain cool enough not to emit electrons; since electron emission by the grid is both



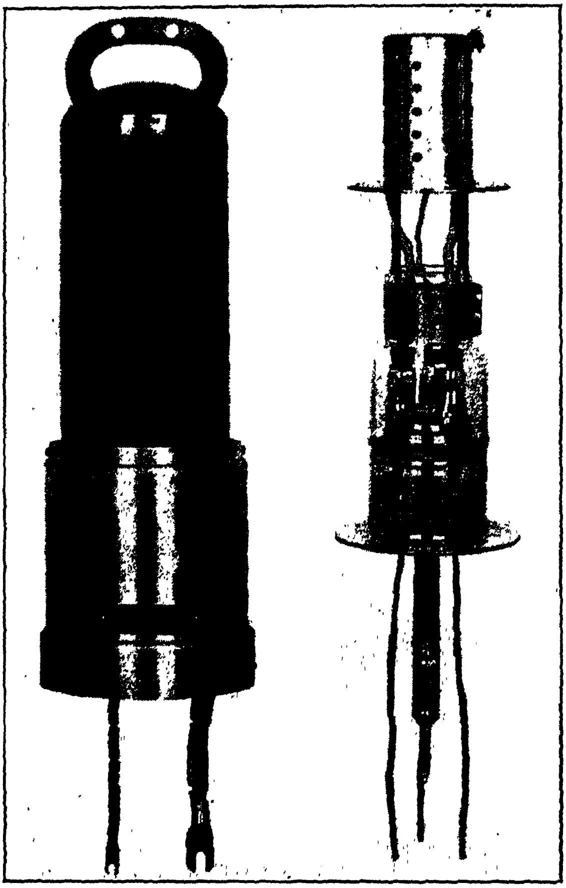
a *b*

FIG. 8.—*a*. THYRATRON FG-41, RATED AT 12½ AMPERES AVERAGE, 15,000 VOLTS MAXIMUM. *b*. CATHODE OF FG-41; CELLULAR STRUCTURE SHOWN IN SECTION ABOVE.

more serious than in the Pliotron, and more likely to happen, because of the larger current. The FG-17 is designed to furnish and control an average current of ½ amp., at any voltage up to 2500. The gas needed for the positive ions is furnished in this, and in the other Thyratrons shown, by a drop of mercury located in a cool place at the bottom of the tube, giving a vapor pressure between 0.005 and 0.020 mm. Thyratrons

containing argon at 0.050-mm. pressure are available for low voltage applications.

The Thyratrons shown in Figs. 7 and 8 embody a new feature, the cellular cathode. This cathode (Fig. 8*b*) takes full advantage of the low resistance of electron flow in ionized gas, by not only removing the elec-



a *b*

FIG. 9.—*a*. THYRATRON FG-53, RATED AT 100 AMPERES AVERAGE, 1500 VOLTS MAXIMUM. *b*. CATHODE OF THYRATRON FG-53.

tron emitting surface far from the anode, but also by locating it on the inside of long, narrow cavities. Electrons easily escape from these cavities, with the help of positive ions, but only a small amount of heat escapes, as compared with that radiated by a filament of equal area. Hence these cathodes are more than 10 times as efficient as filaments. They require about one watt of heat per ampere of emission, to keep them at operating temperature.

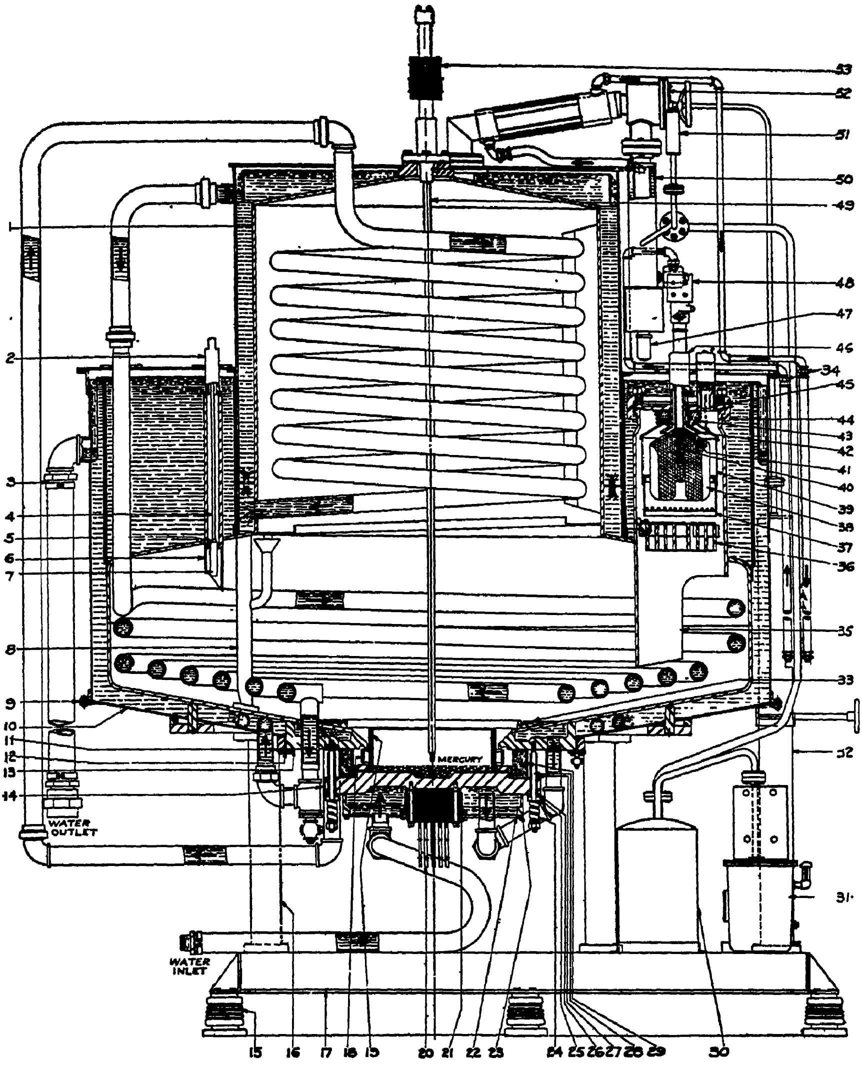


FIG. 10.—METALLIC-TANK MERCURY ARC RECTIFIER, VERTICAL SECTION.

1. Vacuum chamber
2. Excitation-anode seal
3. Overflow graphite nipple
4–7. Excitation-anode: (4) stud; (5) stop nut; (6) shield; (7) tip
8. Mercury drain strut
9. Cord
10. Tank water jacket
11. Tie-ring outer gasket
12. Tie ring
13. Tie-ring inner gasket
14. Cathode
15. Base insulator
16. Leg
17. Rectifier base
18–24. Cathode: (18) outer shield; (19) inner shield; (20) stud; (21) bottom-plate inner gasket; (22) bottom-plate outer gasket; (23) bottom plate; (24) spring
25. Steel washer
26. Insulating washer
27. Supporting stud insulating tube
28. Cathode supporting stud
29. Cathode insulator
30. Gas receiver tank
31. Rotary vacuum pump
32. Vacuum gage
33. Trash rack
34. Carbon-tip nipple
35–41. Main-anode: (35) shield; (36) baffle; (37) grid; (38) tip; (39) grid support; (40) stud; (41) stop nut
42. Cone baffle
43. Insulator
44. Clamping ring
45. Main-anode space ring
46. Grid and main-anode seal
47. Mercury-condensation-pump heater
48. Main-anode terminal
49. Ignition-anode rod
50. Mercury-condensation pump
51. Mercury trap
52. Accordion vacuum valve
53. Ignition-anode coil

The FG-57, shown in Fig. 7, furnishes a current of 15 amp. maximum or 2.5 amp. average, at any voltage up to 1000. The FG-41 (Fig. 8) is rated 75 amp. maximum, 12.5 amp. average, 15,000 volts maximum.

Fig. 9 shows the largest hot-cathode Thyratron available, designated FG-52. It has a cellular cathode (Fig. 9*b*) of slightly different form. It is rated 600 amp. maximum, 100 amp. average, 1500 volts maximum.

The largest Thyratron available, Fig. 10, is of the mercury arc type, with a pool of mercury as cathode. The anodes and grids are placed in side arms, with shields and baffles to protect them from mercury spray and from each other. Each of the 12 anodes is capable of carrying 4000 amp. maximum, and the whole tank has a continuous current rating of 5000 amp. direct current, with overload capacity up to 14,400 amp. for 1 min., at direct-current voltages up to 1500.

Industrial Applications of Thyratrons

The largest field of application for thyratrons, at the present time, is as power amplifiers for controlling mechanical operations. For this purpose the anode is fed with alternating voltage, and the grid controlled by either direct-current or alternating-current voltage, according to the type of operation desired. When alternating voltage is used for the grid, if the grid voltage is in phase with the anode voltage the current starts at the beginning of every cycle, as soon as the anode voltage becomes positive, and stops at the end of the half cycle when the anode voltage reverses. These half-cycle pulses of current are full value, limited only by the load resistance. If the grid voltage becomes positive 90°, or ¼ cycle, later than the anode voltage, the current starts at the middle of each cycle and has only ¼ cycle to flow. The average value of these quarter-cycle pulses of current is obviously only one-half as great as that of the half-cycle pulses. If the phase of the grid voltage is still further retarded, the pulses become shorter and shorter, reaching zero when the grid voltage lags 180° behind the anode voltage. Thus by varying the *phase* of the grid voltage one obtains a smooth variation of *average* current, from maximum value to zero.

This phase-control method is used, for example, in the dimming of theater lights. The master hand turns the knob of a small phase-shifting Selsyn or rheostat, and the lights dim, fade, and blend at his will. In this case the Thyratron current controls the lamps by saturating reactors in series with the lamps, thus varying the impedance in the lamp circuits, and hence the current through the lamps. The lighting in the Chicago Civic Opera House, in the Earl Carroll Theater in New York, and in several RKO theaters, is thus controlled.

Spot-welding illustrates the on-or-off type of Thyratron operation. For this purpose the magnitude of the grid voltage is varied, rather than the phase. A synchronous timing device applies voltage to the grids

according to an exact schedule, say three cycles on, seven cycles off; and the Thyratrons pass current for the whole of each cycle in which the grid voltage is on, but no current when it is off. The passing of current acts as a short circuit on one winding of a series transformer, thus varying the impedance of the other winding, which is in series with the primary of the welding transformer.

The Thyratrons in this welding application simply perform the service of a contactor, but a contactor of unprecedented quality; inertialess, accurate, powerful but controllable by minute power, wearless. It is like the replacement of horsepower with motors. Those who can still remember horses must sense a peculiar thrill in the quiet purr of the motor, in place of the sympathy-arousing perspiration of straining muscles. There is much the same contrast between the sputtering of contactors and the noiseless commutation of the Thyratron.

These two types of control—control of the *phase* of grid voltage, when smooth variation of average output current is desired, and control of the *sign* or *amplitude* of grid voltage for on and off operation—are the basis of most of the industrial service applications of Thyratrons. The list of these applications is already long, and includes wire-drawing, where the Thyratron controls the speed of re-reeling so as to maintain constant tension; synchronizing conveyors in the processing of sheet rubber; counting automobiles, theater patrons, refrigerators coming from production; opening dining-room doors at the approach of a waitress; piling bags and other production articles; conveying products to predetermined destinations; dispatching mail and parcels; turning on and off lights in response to daylight conditions; cutting white-hot steel bars to exact lengths; folding paper napkins; cutting printed wrappers in register, for automatic high-speed package wrapping; sorting beans, at the rate of 40,000 lb. per day, automatically throwing out all that are imperfect or discolored. In many of these applications the photoelectric tube acts as the brains, giving orders, in the form of grid voltage, to its power team mate, the Thyratron.

Power Applications of Thyratrons

In another field Thyratrons promise new services, more spectacular, because more radical and in larger units, though not necessarily more important, than the small-scale but numerous industrial control applications. This field is power conversion. Thyratrons cannot generate electric power, but they are ideally adapted to the task of converting it from one form to another, which they accomplish without motion, noise, or wear. These transformations include changing direct current to alternating (inverter operation); alternating current to direct (rectifier); direct current at one voltage to direct current at another voltage, higher or lower (direct-current transformer); alternating current at one frequency to another frequency, *e. g.*, 60 cycles to 25 cycles or vice versa (frequency

changer); correcting power factor (static synchronous converter); and replacing commutators on motors. In this last application Thyratrons not only offer an ideal solution of the commutator problems of over-commutation, speed limitation, arcing and wear, but also offer new motor characteristics, since the Thyratron commutators can be made conducting or nonconducting at will by controlling their grids.

These power applications, involving large and expensive units, are necessarily slow in development. The only installations thus far are of a laboratory nature. They include a 400-kw. rectifier and inverter, taking power from an 11-kv. 40-cycle alternating-current line, rectifying it to 12,500 volts direct current and inverting this to 60-cycle alternating current, which is used to run a 400-kw. rotary converter; a 500-kva. rectifier-inverter unit operating at zero power factor on a 4000-volt alternating-current line, acting as a synchronous condenser; a 3000-kw. mercury-cathode Thyratron, taking 3000-volt, 60-cycle power and converting it into 25-cycle power; and a 400-hp. synchronous motor operating at variable speed, with Thyratron commutators.

The Phanotron

There are two types of gas-filled, arc-discharge rectifiers, whose electron emission is furnished by a hot cathode. One is the well-known Tungar; the other the new Phanotron.

In the Tungar rectifier the large tungsten filament, used as cathode, is made to yield an abnormally high electron emission, about 10 times as much as in high-vacuum tubes, per unit of area, by heating it to an excessive temperature. Enough argon gas is added to prevent evaporation of the filament at this high temperature. It is found that the gas pressure needed to give this protection is above 1 mm. of mercury, and that the life of the filament improves with increase of pressure up to about 5 cm., which is the pressure used in the Tungar.

This high pressure in the Tungar imposes two limitations: the *voltage* that can be rectified is low, because the sparking potential of argon at these pressures is only 200 volts, and the *current* is limited by the concentration of the arc, which tends to overheat the filament. These limitations apply to the whole range of useful pressures, from 1 mm. to 10 cm.

Both of these limitations are absent at very low gas pressures, between 0.005 and 0.05 mm., and this amount of gas is found to be sufficient to furnish the required positive ions, which are needed to neutralize the electron space charge. The life of the cathode, however, is very short at these low pressures, if it is operated at Tungar temperature; and if operated at normal temperature its efficiency is unsatisfactory.

This problem has been solved, in the Phanotron, by providing a cathode of ample size, so that it can be operated at "normal" temperature and still furnish the required emission; and by constructing it so that its

heat loss is small, taking advantage of the fact that electrons in an arc discharge can go around corners, while heat radiation must go in straight lines. Examples of such cathodes, which are also used in hot-cathode Thyratrons, have been shown in Figs. 7 to 10.

This combination of heat-shielded cathode and low-pressure gas appears to give the four desired characteristics of a rectifier; that is, high

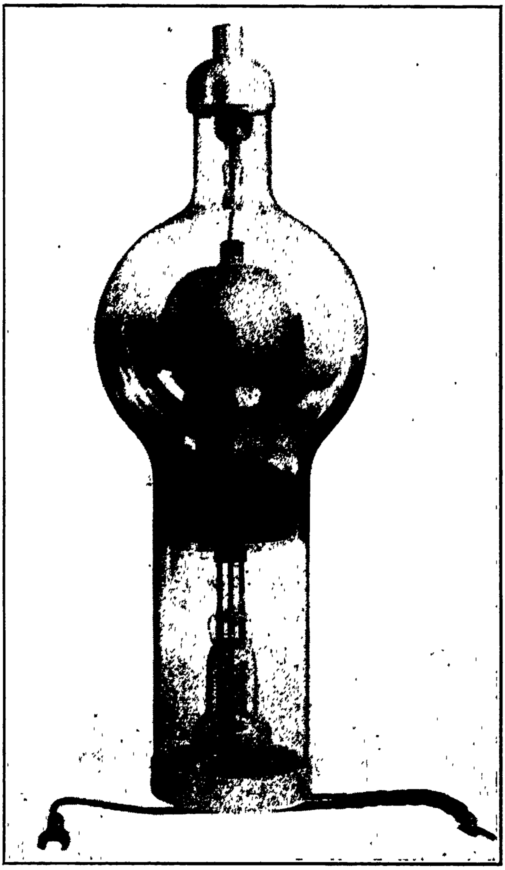


FIG. 11.—PHANOTRON FG-15, WHICH RECTIFIES AN AVERAGE CURRENT OF 10 AMPERES, AT A MAXIMUM INVERSE VOLTAGE OF 20,000.

voltage, large current capacity, high efficiency, long life. Two examples, the FG-15 and FG-52, will serve to illustrate these characteristics.

The FG-15 Phanotron is shown in Fig. 11. Its cathode, which is similar in structure to that of the FG-41, is heated indirectly by a tungsten filament operated at 5 volts, 37 amp. This tube is rated to withstand 20,000 volts peak inverse voltage, and to deliver a maximum current of 40 amp., or a continuous average of 10 amp. The internal voltage drop is approximately 10 volts.

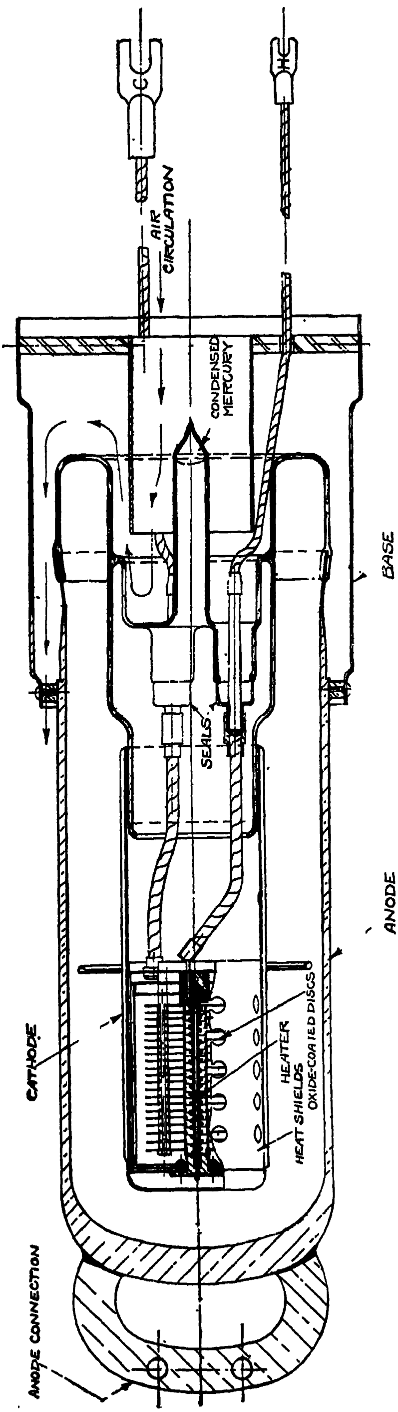


FIG. 12.—PHANOTRON FG-52, DESIGNED TO RECTIFY A CURRENT OF 600 AMPERES MAXIMUM, OR 100 AMPERES AVERAGE, AT ANY VOLTAGE UP TO 1500.

An important application of the FG-15 Phanotron is furnishing 20,000 volts direct-current power for the 100-kw. Pliotron.

The FG-52 Phanotron is a metal tube, similar in external appearance to the FG-53 Thyratron (Fig. 9). Its internal structure (Fig. 12) consists of a multicellular cathode closely surrounded by a copper anode, which forms the envelope. This tube requires 300 watts to maintain the cathode temperature, and is rated at 600 amp. maximum output, 100 amp. average, at any inverse voltage up to 1500. The internal voltage drop, at full load and normal mercury temperature of 60° C., is 6.5 volts.

An interesting application of the FG-52 is as a rectifier at 250 volts direct current, to furnish power for electrochemical plants, Edison three-wire networks, etc. This is a field which rectifiers have been unable to serve heretofore, the mercury arc rectifier because of its poor efficiency, the Tungar because of its low voltage and small current.

Both the Phanotron and the Thyratron are capable of larger current and higher voltage. Life will not be known until more tubes have been in service. The present "expectation" is one year, with the theoretical limit many times this value. The optimum life will probably be determined, as in lamps, by the balance between efficiency and renewal cost.

Sand Filling through Pipes and Boreholes

By Lucien Eaton, Boston, Mass.

(New York Meeting, February, 1932)

The use of filling in mines is less common in the United States than it is in Europe, where in some places it is required by law. In most cases the filling is placed by hand, and the material used for this purpose is the waste from development and that sorted from the ore in the stopes. When these sources provide an insufficient supply, additional rock must be quarried on surface or mined underground, or some other material must be used. In different parts of the world sand, clay, gravel, culm, granulated slag, mill tailings, and even ashes, have been used as filling material in place of or in conjunction with rock.

Of all the materials used for filling the best is probably clean sand. Gravel, finely crushed rock, granulated slag and coarse mill tailings are almost as good, and the methods used for placing and confining sand will apply equally well to any of these. The advantages of sand are its comparatively slight compressibility and the ease, economy and convenience with which it may be placed. On account of the voids, which necessarily occur in fills made of coarse waste, such fills will compress 10 to 25 per cent before they develop their full strength, whereas clean sand seldom compresses more than 5 per cent. Sand is especially desirable when practically complete filling is required, and sand derived from mill tailings, especially when not thoroughly cleaned, has been remarkably effective in sealing off and extinguishing mine fires. When natural sand is available on surface, or when it can be obtained from the mill tailings or tailings dump, the cost of using this material for fill will probably be considerably less than that of rock.

In this paper it is not proposed to discuss all methods of filling, but to describe some of the methods used for placing sand fill and to make a brief comparison of their merits.

Methods of Placing Sand

Sand may be transported and placed underground by several methods, which may be classified as follows:

1. *By Hand.*—The sand, which must be comparatively dry, is taken down the shaft in cars on cages and is distributed to its destination in these cars, or it is dumped down raises and is drawn off and distributed by means of cars, wheelbarrows or scrapers.

2. *By Water.*—The sand is mixed with water, and is flushed down pipes or boreholes, and is distributed by pipes or launders. The material so distributed may be as low as 20 per cent or as high as 75 per cent solids, depending on the size of the particles and the means of distribution.

3. *By Compressed Air.*—Dry sand is dumped down raises or large pipes, from which it is drawn off and distributed through smaller pipes. Two systems are in vogue:

(*a*) An intermittent high-pressure system, in which the sand is charged periodically into a tank, from which it is blown through pipes to its destination by compressed air at a pressure of four to five atmospheres.

(*b*) A continuous low-pressure system, in which the sand is fed into a pipe through which a large volume of air, at a pressure of about one-half an atmosphere, is flowing.

4. *By a Combination of Compressed Air and Water.*—A mixture of sand and water is sent down through pipes or boreholes, and is distributed through pipes, into which small jets of compressed air are introduced at intervals.

The cost of placing fill by hand is usually high, except in places where the wages of unskilled labor are very low, and it is always difficult to make a complete fill. The upper levels of the Aragon mine at Norway, Mich., were filled with sand over thirty years ago, and the sand was distributed with scrapers, (probably the first use of scrapers underground), but the system was abandoned on account of its cost.

Flushing culm into the old workings of anthracite mines was practiced in Pennsylvania between 1860 and 1870,[1] and is common practice at these mines today. Hydraulic methods of filling have been used in many parts of the world, and a few examples of present practice will be described. One of the chief disadvantages of the hydraulic method is the cost of pumping back to surface the water used for transporting the sand. In a shallow mine this may not be a serious item, but its importance increases with depth. Another important item of expense is the wear on the pipes used for transporting and distributing the filling material. This wear may be reduced to a small amount, if special pipes are used, but the first cost is very much larger. In any installation the cost of moving the pipes or launders is an important item of expense.

Compressed air has been used successfully for placing dry sand-fill in this country, and the practice is said to be gaining favor in Europe. Coarse material is handled successfully, the diameter of the largest particles being one-third of the diameter of the pipe through which they must pass. The principal disadvantages of the method are the cost of compressing the air and the frictional wear on the pipes, especially at

[1] D. C. Ashmead: Mining of Thin Coal Beds in the Anthracite Region of Pennsylvania. U. S. Bur. Mines *Bull.* 245 (1927) 99.

the turns. The low-pressure system costs less for air compression but uses larger pipes.

The combination of compressed air and water is in use in Cumberland, England, and suffers, of course, from the same disadvantages as the hydraulic and compressed air systems, but not to as great an extent as either of them. The use of compressed air makes it possible to reduce the percentage of water in the mixture, thereby reducing the pumping cost, and it also makes possible the distribution of material through long lateral pipes, and prevents the settlement of sand in low spots in the line. These advantages more than offset the cost of the compressed air.

Mill Tailings as Fill

In 1923, H. J. Rahilly described the methods followed at Butte, Mont., for extinguishing mine fires in stopes.[2] The material used was thickened pulp from the mills, and the consistency varied from 18 to 30 per cent solids. By screen test, 29.90 per cent remained on 150-mesh and 47.90 per cent on 200-mesh. This material was therefore half slimes, and was eminently suited to the purpose for which it was used. The pulp was sent down the shaft through an 8-in. cast-iron pipe, and was distributed on the level in 4-in. cast-iron and wrought-iron pipes. Sharp curves were avoided. Pressures as high as 500 ft. were used, and it was found that the pulp could be transported 800 ft. on the level for every 100 ft. of head in the column-pipe. Occasionally this figure could be exceeded. When the pulp was distributed to the stopes in launders, a grade of 2 per cent was satisfactory.

The high percentage of fine material in the pulp increased the ease of transportation and reduced the frictional wear on the pipes, but it slowed up the drainage and made necessary the building of substantial masonry dams in the drifts and raises below the stopes that were filled.

The practice used in filling stopes with sand derived from mill tailings at Minas de Matahambre in Pinar del Rio, Cuba, has been described by George I. Richert.[3] This information has been supplemented by D. D. Homer[4] in the past year.

The mining system used is horizontal cut and fill in large irregular pipes of ore of lenticular cross-section, dipping about 45°. Mill holes and a ladder road are cribbed up through the fill, and are covered on the outside by 10-oz. burlap. When a height of 70 ft. is exceeded, it is necessary to line the chutes with plank, because the lower burlap rots out.

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[2] H. J. Rahilly: Mine Fires and Hydraulic Filling. *Trans.* A. I. M. E. (1923) **68,** 62 et seq.

[3] G. I. Richert: Filling Stopes with Mill-Tailing. *Eng. & Min. Jnl.* (1929) **127;** also U. S. Bur. Mines *Inf. Circ.* 6145 (1929).

[4] D. D. Homer: Rubber Pipe Lining Minimizes Pulp Abrasion. *Eng. & Min. World* (1931) **2.**

boreholes must be always emptied, when operations cease even for a short time, and the amount of water in the pulp is regulated so as to maintain an even flow through the hole.

Costs are given as 6.6d. to 7.5d. per ton of sand placed, and as 4s. 4.75d. per square fathom for a 52-in. stoping width. This is $0.185 per cubic yard, or about $0.10 per ton. It does not seem possible that the cost of pumping the water back to surface is included in this figure.

Filling Stopes with Sand

Coarse, dry stamp sand up to ⅜-in. in dia. and containing no slimes was used to supplement as filling the waste rock sorted from the ore in the stopes of the Champion mine of the Copper Range Co. at Champion, Mich. The sand was brought back from the mill in railroad cars, and was dumped down raises, from which it was drawn off through a short 10-in. pipe into a tank and was blown into the stopes by high-pressure air.[7] The 10-in. pipe was equipped with a quick-acting gate, and discharged directly into a steel tank, holding 1½ tons, which had a self-closing door in the top. The tank was cone-shaped at the bottom, and terminated in a 6-in. tee, through one side of which the sand was discharged. Through a plug in the other side of the tee a 1-in. pipe with a ¼-in. nozzle entered. When the tank was nearly full the door in the top was closed, compressed air at 70 lb. per sq. in. was turned on above the sand and also in the small pipe in the tee. The air jet at the bottom blew the sand away through a 4-in. pipe, and the pressure above forced the charge down into the tee. By this means the sand could be blown through 250 ft. of 4-in. pipe and 20 ft. beyond the end of the pipe.

One man was required to operate the tank, and could fill and empty it in 2 to 3 min. The compressed air required varied from 500 to 1500 cu. ft. per minute, and the average cost for air was about 2¢ per ton. The capacity is about 30 tons per hour. At this mine 4-in. extra heavy pipe in 6-ft. lengths with Dresser couplings was used, and had a life of 3 to 4 months. The absence of threads on the pipe nearly doubled its life. The greatest wear was in the elbows at the turns, and this was finally overcome by using a long-sweep elbow made of a special hard iron. This method is no longer in use because a change in the mining system has made additional fill unnecessary.

Pneumatic Process

In July and September, 1931, two interesting articles were published by Henry A. Dierks.[8] In the article on pneumatic stowage is a descrip-

[7] W. H. Schacht: Mining Methods of the Copper Range Co. *Trans.* A. I. M. E. (1925) **72**.

[8] H. A. Dierks: Hydraulic Back-filling. *Coal Age* (1931); Pneumatic Stowage. *Ibid.*

tion of the low-pressure pneumatic system as used in German coal mines. A central storage bin is provided underground, and to this bin the waste from development is trammed, and from surface is brought such suitable material as is available, such as washery refuse, ashes or sand. All of this material is passed over a screen, and the oversize is crushed.

From the lower end of the bin the filling material is fed by a disk feeder into the pipe line, in which a continuous stream of air at 8 lb. per sq. in. is passing. Close regulation of the feed is necessary to prevent waste of air or overloading and clogging of the pipe.

In a 10-in. pipe pieces of slate of 3-in. dia. can be blown without difficulty 1500 ft. horizontally, and further if the grade is downhill. The capacity is usually 30 to 40 cu. yd. per hour, and the fill produced is quite as satisfactory as that deposited with water. Wear on the conveying pipe is not as heavy as with the high-pressure system, but the quantity of air required is large, being about 300 times the volume of the material transported. The power cost is the most important operating expense, about 6 hp. of connected load being required for compressor, crusher and feeder, per cubic yard of hourly capacity.

Sand Placed by Water and Compressed Air

At the Hodbarrow mine, at Millom in Cumberland, England, beach sand, placed by a combination of water and compressed air, is being used as filling. This great mine has been in operation over 100 years, and has produced over 22,000,000 tons of ore. The ore is a hydrated hematite and is overlain by sand and other loose material. It is on the seashore, and the area that has been undermined is protected from the ocean by a sea wall. Most of the ore has been mined by top slicing, but the difficulties brought on by subsidence became so serious that it was desirable to change the system of mining. The process has been reversed, and the ore is now mined from the bottom upwards, the slices being filled instead of being caved.

A main haulage drift is driven in the footwall about 15 ft. under the ore, and chutes are built and vertical raises extended up into the ore at 60-ft. intervals. From these raises crosscuts are driven right and left to the limits of the block that is to be mined, and the raises are connected by a drift. From the end of each crosscut a drift is driven right and left halfway through the pillar, and, when these have been completed, the crosscut is dammed off with plank, behind which brattice cloth is hung. Brattice cloth is also hung along the sides of the drift on which there is still ore to be mined, so as to prevent contamination of the ore by the filling. Sand mixed with water is then blown in, and the opening is filled. There is difficulty in filling the space above the bottom line of the caps, as all the workings are timbered.

When the first drift has been filled, another drift is driven alongside it, and this is filled in turn. The procedure is repeated, until the side of the pillar that is left to protect the haulageway has been reached, which is about 20 ft. on either side of the center line. The crosscuts are then filled back to the raises, the raises are extended one slice higher, about 9 ft., and another slice is taken in the same way as before. The pillar over the main haulageway is mined last, and each raise, when the ore around it has been mined, is closed with a concrete dam.

As the price received for the ore is dependent upon its analysis it is essential that as little sand as possible be mixed with the ore. Careful blasting is necessary therefore, and, although the brattice cloth is punctured frequently, the sand stands so well that there is very little contamination. There is also another reason for confining the sand very closely. The main haulage is operated by an endless rope, which is frequently in contact with the floor of the drift, and, if sand is washed out into the drift, the rope is quickly cut to pieces. No trouble is usually experienced on this score, as the sand normally settles to the bottom and the water drains off the top, but occasionally the sand does not settle and pack well and the water drains out along the bottom, carrying sand with it.

The filling material is sent into the mine through 4-in. pipes with Victaulic joints. The pipes are hung in boreholes or in disused shafts. At the collar of each pipe there is a concrete bin or hopper, and over the opening of the pipe there is a wire screen about 1 ft. square with 1-in. openings. If the sand is near enough to the borehole, it is flushed into the hopper with a hose. Most of the sand now used, however, is loaded into cars with a clamshell bucket, and is dumped into the bin, from which it is washed into the pipe by a 2-in. hose with a 1-in. nozzle. The stream normally carries 40 per cent solids and 60 per cent water, but sometimes this percentage is reversed, or the percentage of solids may fall as low as 30 per cent. The sand grains are all rounded, and, for this reason apparently, the wear on the pipes is not serious.

The feature of the system is the use of compressed air as a booster. The pipe has long-sweep elbows, and into some of these ¾-in. pipe is tapped on the center line of the pipe on the exit side, so that the air jet will not impinge upon the wall of the pipe. The ¾-in. pipe is reduced to ⅛-in. dia. at the end. The compressed air increases the velocity of the stream, and keeps the pipe clear of sand without regard to the gradient. This makes is unnecessary to remove sags and low places in the line, and the velocity of the discharge helps in filling the upper portions of the stope.

A signal cable is strung through the borehole so that signals can be given for the control of the sand.

The Victaulic couplings are easily and quickly connected, and allow slight bends in the line. It should be noted, however, that the English

Victaulic joint has a rolled collar on the end of the pipe, instead of the cut groove of the American joint.

As the free discharge of air at 70 lb. through a ⅛-in. nozzle is given as only 19 cu. ft. per minute, the air consumption is not a heavy item, and as the mine is less than 600 ft. deep the pumping charge for the water from the fill cannot be large. No data are available as to the cost per cubic yard or per ton.

Local Conditions Govern Choice of Method

From the foregoing it is apparent that the choice of the proper method to be used in placing sand fill must depend upon local conditions. If the depth is not great, and if there is ample pumping capacity, the hydraulic method would probably be best, assisted by compressed air in long laterals. Apparently, also, water is necessary for moving sand through boreholes, but, if the mine is deep, the quantity should be kept as low as possible, and a thick pulp may require the assistance of compressed air, if the grades available are not steep enough to permit the use of launders.

In a deep mine, where the pumping charge would be excessive, the pneumatic method would be preferable, especially if coarse material was to be used.

The size of the pipe depends on the size of the material to be transported and upon the cost of the pipe. A certain velocity must be maintained in order to keep the pipe from clogging, but too high a velocity means excessive wear. At best it is a compromise between first cost and maintenance.

Under the most difficult conditions the best system would seem to be a comparatively thick pulp transported through rubber-lined pipe with pneumatic boosters at suitable intervals.

DISCUSSION

(Scott Turner presiding)

J. H. Pierce, New York, N. Y. (written discussion).—Mr. Eaton lists four methods of placing the sand filling in the mines as follows: By hand, by water, by compressed air, and by a combination of compressed air and water.

There is another method of back filling which Mr. Eaton has neglected to mention, a method used extensively in the potash mining in central Germany. In this district back-filling material may consist either of sand or of crushed rock salt which constitutes the tailings from the potash operation, or a mixture of sand and salt.

The filling mixture is dropped either through a pipe line in the shaft or through a borehole, on to a shaking conveyor which conveys it to the working chamber to be filled. In this working chamber a mechanical device is set up which is somewhat similar to a mining fan. The conveyor delivers the material to the periphery of the fan and it is then distributed to the face by the centrifugal action of the fan wheels.

The vein being mined is 30 ft. or more thick, but under the German laws only 10 ft. of the vein may be extracted in one cut. After this 10-ft. opening has been backfilled, the second bench is taken. Thus, the mechanical device has to pack an area approximately 10 ft. high by 30 ft. wide at one operation. This method eliminates the hand labor formerly employed in transferring the sand from the conveyor into the pack and, in addition, procures much more efficient back filling.

The Sublevel Inclined Cut-and-fill Stoping System

By Albert Mendelsohn,* Painesdale, Mich., and Charles F. Jackson,† Washington, D. C.

(New York Meeting, February, 1932)

The system of stoping described in this paper was first introduced at the Champion mine of the Copper Range Co., Painesdale, Mich., in 1929, and since that time has been developed to a high state of efficiency. The method was devised to overcome some of the objections to the system of horizontal cut-and-fill stoping formerly employed at this mine and has done so successfully. The objections to the old system became more pronounced as the depth of mining operations increased and a change became a matter of necessity. While the combination of conditions encountered at the Champion mine is rather unusual, it is believed that the sublevel system developed there could be employed advantageously at other mines, though certain modifications might be necessary. This paper describes the method and the results obtained through its use, and suggests modifications for making it applicable under quite different conditions from those at the Champion mine.

Champion Lode

The lode being mined at the Champion mine is the heavily brecciated top of what is known in the district as the Baltic lava flow. The brecciated portion of the lode varies from 6 to 50 ft. in width. Under this breccia is the thinner, tighter, amygdular portion of the lode; under the amygdular portion is the trap footwall. Native copper is found deposited in the rock fragments of the breccia, in the cementing material of the breccia, in the amygdules and in the trappy rock under the amygdular part of the lode. In some places masses of copper many tons in weight extend far into the footwall of the lode.

The Baltic lode on Champion property is 8000 ft. long, very straight along the strike, and dips uniformly 70° to the west for 3000 ft. in depth, at which point the dip begins to flatten slightly. The width of the copper-bearing portion of the lode varies from 10 ft. to as much as 80 ft. in a few places; the average width now mined is 17 ft.

The lode rock is hard—the standard one-man drilling machines used in the mine drill from 10 to 14 in. per minute when mass copper is not

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† Principal Mining Engineer, U. S. Bureau of Mines.

encountered. The hanging wall, which is the bottom of another flow and shows blocky structure, is full of minute seams, running apparently in every direction, with larger seams running parallel to the strike and dip. When exposed over an area of only a few square yards and unsupported by timber or fill, the hanging wall will begin to disintegrate and fall into the stope. There is no definite foot slip; the lode becomes more and more trappy on the foot side the deeper it goes into the flow. The wide portions of the lode seem to be due to large accumulations of fragmental material on the very top of the flow. In some places the lode is split into two branches, separated by very trappy, amygdular rock.

Copper is not uniformly distributed in the lode but in a large way it follows chutes pitching from the north down toward the south. The copper may occur on the hanging side, in the middle or on the foot side of the lode; or it may be distributed clear across the lode from the hanging wall to the footwall. In a smaller way, copper occurs in patches surrounded by barren rock, which are distributed irregularly both along the strike and on the dip as well as across the lode. Thus the absence of copper in the face at any point does not signify its absence ahead of, above, or to one side of the opening.

The rock shipped from the Champion mine during 1931 ran 41.5 lb. of refined copper per ton. This grade of material was obtained by hand-sorting broken rock of an average tenor of 22.5 lb. per ton. It is estimated that before selection of stoping ground, all the lode rock opened by development would average only 9 lb. of copper per ton.

Mining Conditions and General Development

The conditions affecting mining practice and costs are essentially as follows:

1. A lode persistent along the strike, but varying greatly in width, with zones of barren or very lean rock separating the workable ore chutes in which the copper is in turn irregularly distributed across the lode.
2. Hard lode material lying beneath a seamy and often badly broken hanging wall which is more treacherous and harder to hold the deeper mining operations progress.
3. A low-grade ore in which copper occurs in the native form and is readily distinguished from barren rock, so that sorting of waste from ore in the stopes is feasible; about 40 per cent of the rock broken is sorted out as waste.
4. Present stopes are in areas lying below old workings which have been filled with sand or waste rock and from which a plentiful supply of waste for filling purposes is readily available. The drawing of waste from the old stopes is permissible, since any caving or subsidence brought about thereby will not endanger surface improvements, shafts or other active workings.

General development consists in cutting shaft stations at 100-ft. intervals, driving short crosscuts to the lode and driving haulage drifts in the lode to the boundaries. These drifts are 8 ft. high and 9 to 12 ft. wide. Stope development is described in the following paragraphs.

Stoping

Old Method and Reasons for Change of Method

The ordinary horizontal cut-and-fill stoping method with hand sorting of waste in the stopes was employed from 1905 to 1929. This earlier practice was described in detail by Schacht in 1923.[1] The method of stoping was changed late in 1929 in an effort to reduce stoping, maintenance and haulage costs. As the mine became deeper the average grade of the ore decreased, and at the same time, owing to depth and the large proportion of the lode removed by stoping, the remaining ground gave evidence of being unable to withstand the weight of the hanging country. Spalling off of the stope backs was often followed by heavy falls of ground, rock bursts became increasingly frequent, and the long, wide, horizontal stopes were difficult to keep open.

Under the old system of horizontal stoping on the advance (beginning near the shaft and advancing toward the boundaries), timbered drifts had to be maintained for a considerable length of time under mined-out blocks of ore. Maintenance of these drifts was expensive and frequent retimbering was required.

The foregoing indicates in a general way the conditions leading to a change in the stoping method. More specifically, the reasons for the change are summed up as follows:

1. The excessive cost of maintaining the horizontal stopes when, to prevent heavy falls of ground on account of crushing, every stope had to be supported from end to end with timber props and cribs, and this support had to be repeated for every horizontal slice mined from the back.
2. Building the ore passes of rock was slow and costly; timber could not be used because it would rot before the floor pillar could be mined.
3. On account of the long life of a level and because of the movement of the ground, induced by the weight of the hanging country, the level walls became distorted, the track bulged upward and the level timbers broke or rotted out. Level maintenance expense became excessive.
4. Filling operations and the building up of ore passes interrupted the production of ore in the stopes; with five stopes on a level, only three could produce at one time because of these interruptions, though all five had to be maintained in order to secure the required output.

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[1] W. H. Schacht: Mining Methods of the Copper Range Co. *Trans.* A. I. M. E. (1925) **72**, 346–370.

5. Although horizontal stoping lent itself nicely to hand sorting, there were numerous patches of ground in which the copper was so uniformly distributed that hand sorting was impossible. In such cases the horizontal arrangement did not permit large tonnages to be handled quickly. It became desirable in such cases, and indeed in all stopes, when the rock had been picked out, to move the copper rock mechanically and as efficiently as possible to the tram cars on the main level. Power shovels and picking belts were tried, but were too unwieldy in the heavily timbered stopes.

Hand Sorting

A feature of the old method which it was necessary to retain under any new system of stoping that might be adopted was the hand sorting of waste in the stopes as introduced at the Champion mine in 1905 by F. W. Denton. The irregular distribution of copper in the lode has already been referred to. It is not uncommon when drifting in copper-bearing areas for the grade of the rock to change from good to poor and vice versa several times in a month. Moreover, in the good areas, the copper occurs principally in the form of small nuggets or masses surrounded by barren rock. In drifting operations, sorting is accomplished by simply loading waste and copper rock into separate cars.

It would be highly undesirable, when stoping, to send to the surface as copper rock all the rock broken, containing as it does over 40 per cent of barren rock. Haulage and hoisting, exclusive of fixed maintenance charges, at present cost 25¢ per ton; preliminary crushing in the shaft house, 6¢ per ton; transportation to the stamp mill, 24¢ per ton; and stamping, 40¢ per ton. This indicates that there is a saving of 95¢ on every ton of barren rock that can be left in the stopes. Furthermore, by eliminating barren rock from the rock shipped from the stopes, its copper content per ton is increased, and all subsequent costs, figured on a per pound of copper basis, are decreased proportionately.

Difficulties due to barren rock in the lode are encountered in all mines in this district in some degree. Some of the narrower mines have met the difficulty successfully by mining around the larger areas of barren rock and leaving these areas as pillars to support the openings, using shrinkage stoping. In the Champion mine, where copper in the footwall or hanging wall is often hidden from the miners by horses of barren rock, it is desirable to give the miners great freedom in the matter of breaking barren rock on either the foot or hanging side of their stopes. This practice has found tens of millions of pounds of copper in this mine. Because the amount of barren rock to be disposed of is increased by this practice, and on account of the physical conditions in the Champion lode, such as width of lode, dip and a bad hanging, it is necessary to employ a filling method. In addition to hand sorting, the sublevel

method of mining now practiced, with sublevels separated by 25 ft. of rock, enables us to mine around the larger areas of barren rock at this mine also.

There is never any difficulty in distinguishing between filling material and freshly broken stope rock. The filling material, because of its original travel through wet raises, and later its long repose in the upper stopes, is completely covered with a mixture of moisture and dust, which gives it a dark red, muddy appearance that contrasts strikingly with the dry, gray appearance of freshly broken vein rock.

In order to aid the picking operation, the rock is broken as large as possible. The "handy" rock is picked according to the simple rule that any rock containing any visible copper must be sent out of the stope as copper rock; any rock in which no copper can be seen must be left in the stope as barren rock. The fines are sent as copper rock, unless the stope is so lean that in the judgment of the pit boss it must be left in the stope. Tram cars are inspected and poor rock found in them entails the penalty of a layoff of at least one day for the pickers responsible. Stopes are inspected, and copper rock discarded as waste calls for a similar penalty. Contract schedules are arranged so that the men can make more money by careful picking than by careless working.

Size and Productivity of Stopes

In the deeper mines of the Calumet and Hecla Co. some of the difficulties due to depth and maintenance were overcome as far back as 1909 by developing well ahead of stoping and doing all mining on the retreat. The logical step seemed to be to follow their example and start stoping at the boundaries and retreat toward the shaft. Their experience showed that a stope could not be more than 100 ft. long; that often with stopes longer than 100 ft. it was impossible to stope out the ground from one level to the next one above before the crushing of the stope back and hanging wall would commence and drive the men from the stope.

Experience with horizontal cut-and-fill stoping, and hand picking in the stopes at the Champion mine, showed that it required 200 ft. for one working place to give continuous production. The production under such an arrangement would be too small for the level. It was necessary to have at least three productive working places on each level, and at the same time mine on the retreat. The solution was to split the 100-ft. level interval into three sections or backs by means of sublevel drifts, giving three points of attack, and to turn the stopes up at an angle equal to the angle of repose of the filling material, just as was done in removing the floor pillars under the old Baltic mining method.

Development in Sublevel Inclined Cut-and-fill Method

Haulage levels are first driven from the shaft crosscuts to the ore boundaries. About 200 ft. back from the boundary a two-compartment

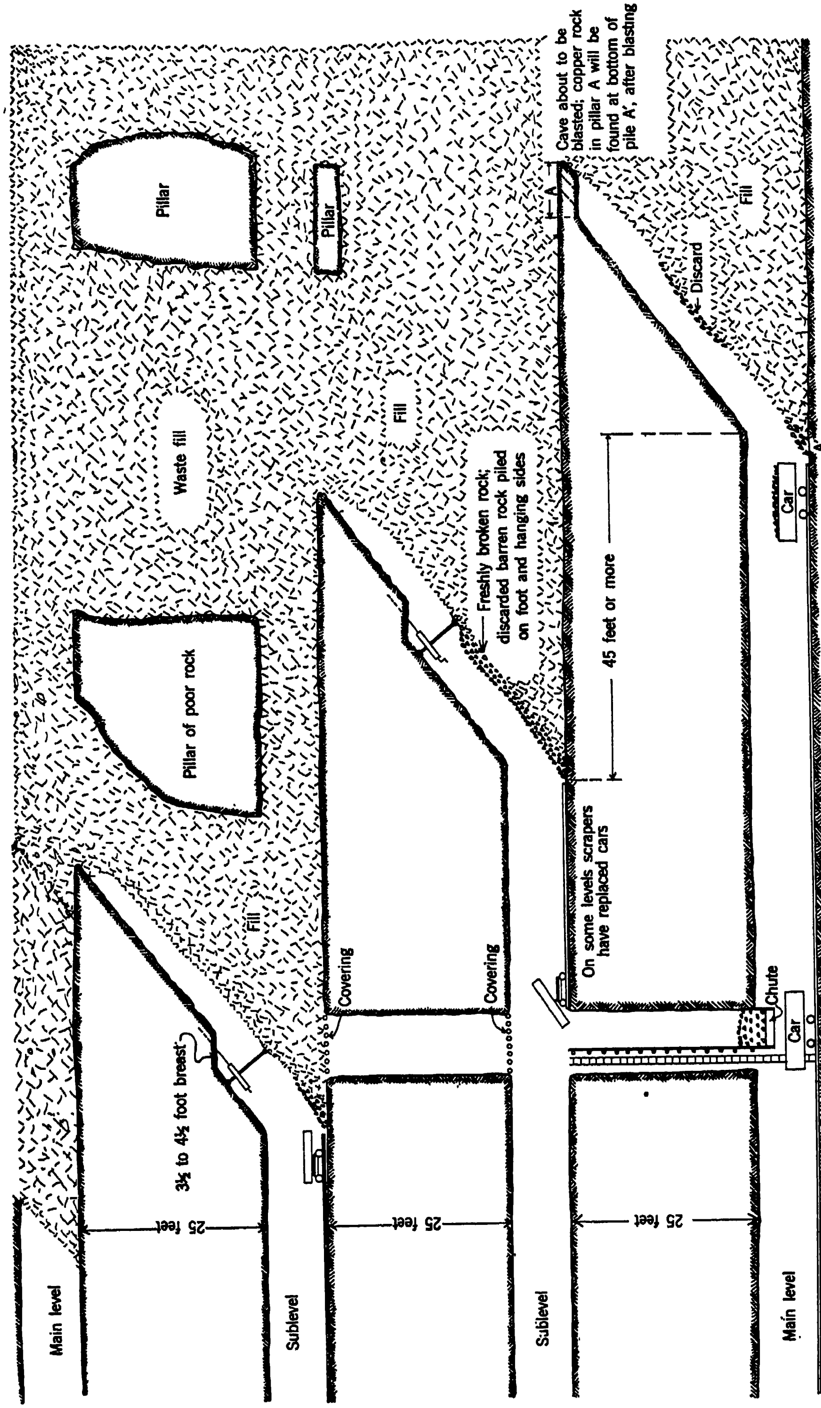

FIG. 1.—NEW STOPING METHOD, SUBLEVEL INCLINED CUT AND FILL, USING CARS.

(chute and manway) raise is driven up to the level above as shown in Fig. 2. Stations are cut in this raise 33 and 67 ft. above the floor of the level respectively, and from these stations subdrifts are driven on the lode in both directions. These subdrifts are driven the full width of the copper-bearing portion of the lode and thoroughly explore both the footwall and hanging wall sides. Scrapers operated by 15-hp. double-drum electric hoists are employed for scraping the broken rock into the chute as the drift advances. The scraper is of the hoe type, 48 in. wide, weighs 750 lb. and has a removable molychrome-steel cutting edge.

Drifting is done by two miners on each shift, using two drilling machines. By means of the scraper, upon coming on shift the miners can quickly muck back for a clean set-up, and while one miner is starting

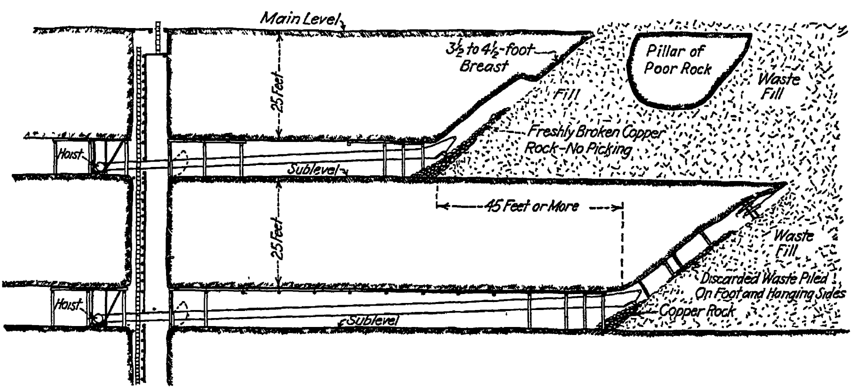

FIG. 2.—NEW STOPING METHOD, SUBLEVEL INCLINED CUT AND FILL, USING SCRAPERS.

the drilling of the next round, the other scrapes the broken rock back to the chute. After completion of the scraping, the latter returns to the face and starts drilling off his part of the round.

When the upper subdrift reaches the boundary, a raise is put up to the level above, after which stoping may be started.

STOPING IN SUBLEVEL INCLINED CUT-AND-FILL METHOD

Beginning at the upper sub and at the boundary raise, the back of ore is stoped out by a series of inclined cuts (Figs. 1 and 2). The miners advance the breast up the slope at an angle of 38° until only a small pillar, 2 to 4 ft. thick, is left supporting the fill in the worked-out stope above. In wide stopes this small pillar must be left 4 ft. thick. The pillar is drilled over and the holes left to stand without blasting until all copper rock has been picked out of the stope and all exploration has been done on the foot and hanging sides. The foot sides of some stopes have been mined to widths of over 60 ft. before the last pillar was blasted out. When all such work has been completed in a stope the pillar is blasted, and the fill from above rushes down and completely fills the

stope. Most of the copper rock contained in the small pillar last blasted is found at the bottom of the pile of fill. The miner now starts again at the bottom of the slope and goes up with another inclined slice. Loose ground is supported on props or cribs as conditions may require.

The miner drills his round of holes parallel to the angle of repose of the fill, using four, five or six holes across the stope, depending upon its width. The burden on these holes is usually 3 to 4 ft. but varies with the nature of the ground. In very lean rock where a considerable discard must be made, the burden is smaller in order to break the rock in a better condition for sorting. In rich rock, where there is not much loose or heavy ground, the burden on the holes is greater. In heavy ground a breast not over 3½ ft. thick is safer because the miners are nearer the back and can more readily bar down or prop any loose ground left after the blast.

In the wider stopes and with favorable ground conditions 60 tons per machine shift can be broken. When indications of copper appear on the sides of the stope (usually the footwall side) only one round is taken beyond the copper indications before rigging up on the foot to determine whether or not the stope must be widened out. If copper continues into the wall, it is followed until the trap rock is reached before resuming stoping up the slope.

In heavy ground, should there be indications of copper in either wall at the bottom of the stope, it is not blasted out until just before the stope is filled. Otherwise disturbance at this point by blasting is likely to make the brow excessively heavy.

Although it is the practice to drive the sublevel drifts the full width of the copper-bearing part of the lode, some very rich "pockets" of copper have been opened in the foot beyond the sides of the drift during the stoping operations. In several places stopes were widened from the original width of 15 ft. to as much as 58 ft. before foot trap was definitely struck. In some of these instances the ground was very heavy, and props had to be placed from the fill against the back to protect the miners and pickers while handling the foot rock.

When approaching the fill above, the miner must alter the depth of his holes to suit the character of the ground in each instance. A pillar is left as thin as possible, so that when it is blasted out only a small amount of copper will be buried when the fill runs in from above. By careful blasting it is often possible to leave a pillar the full width of the stope, less than 1 ft. thick at the fill or inside edge and about 4 ft. thick where it joins the solid back. Under ordinary conditions it is usually possible to leave a small pillar tapering from 4 ft. to less than 1 ft., 3½ ft. long and the full width of the stope. After all copper rock is cleared out of the stope and this pillar is blasted, a large part of the rock from the pillar runs down the stope and is found at the bottom.

When a barren or very lean section of the lode must be left as an unmined pillar, the miners are moved back to the next appearance of copper in the subdrift and stoping is resumed at that point by putting up an inclined raise.

When a stope has been carried back 45 ft. or more, a similar stope may be started on the next sub below.

Handling of Stope Rock in Sublevel Inclined Cut-and-fill Method

The rock is blasted against the fill, and when cars are used as shown in Fig. 1 the copper rock is picked out and is thrown into the car and trammed to the chute. It was at first thought that it would be impracticable to use scrapers for transferring the copper rock to the chute because of the necessity for sorting. Experiments with the scraper soon demonstrated, however, that while scraping was much more efficient where hand sorting was not required, it could often be employed with appreciable savings as compared to car loading and tramming even where sorting was necessary.

In stopes where the rock is handled by means of scrapers (Fig. 2) the stope organization consists of two miners on each shift. The miners pick out the copper rock and throw it to the bottom of the slope, where it lies in the trench formed by the scraper until a sufficient amount has accumulated; then the miners go back to the slusher hoist and scrape it into the raise. When the rock is very rich, or when the copper is very finely disseminated in the rock, hand sorting is not practical and the entire blast is scraped to the raise. In either case, the sheave for the tail rope is so placed, just above the brow of the slope and over the bottom of the rill of the fill, that the scraper cannot pick up filling material.

The two miners do all the work to be done in the stope; drilling, blasting, picking, scraping and timbering. The usual procedure is for the men to be engaged in the same operation at the same time; they both drill, both pick, or are both engaged in scraping, timbering, etc. When the rock is rich enough to be scraped without picking, the usual cycle of operations leaves a considerable space of time, while the miners are drilling, when the stope is unproductive. This has been overcome by introducing a scraper runner and allowing the miners to rig on the fill and drill the breast while scraping is going on beneath them. The usual procedure, however, is to work the stope at all times with two miners.

Engineers measure the volume of the excavation made in each stope monthly to determine the total rock broken and the average width of the stope. Underground records show the number of cars of copper rock shipped from each stope. The time books show the number of shifts of all labor working in each stope. From these data the engineers calculate monthly the bonus to be paid to each party of contractors.

The use of tabulated rate schedules facilitates the work of calculation. The rate schedules cover:

1. The bonus per shift to be paid for a given tonnage of rock broken in a stope of a given width; the wider the stope, the greater the tonnage broken per man per shift for a given bonus.

2. The bonus per shift for a given tonnage of copper rock produced, after sorting, from a stope showing a given ratio of total rock broken to total copper rock produced; the greater the amount of barren rock that must be handled in sorting, the greater the bonus pay per shift for a given tonnage of copper rock produced.

3. The allowance of powder per ton of rock broken for each width of stope.

Advantages Gained by New Method at Champion Mine

The tons produced per man per shift for 1931, including all labor in the stopes, was 43 per cent higher than in 1921, when the mine operated on the horizontal cut-and-fill method. The grade of rock produced after hand picking was slightly higher in 1931 than in 1921. The year 1921 is used because good men were plentiful, just as they were in 1931. Conditions for mining in the stopes were much more unfavorable in 1931 than in 1921. In 1921 the stope backs were solid and had not begun to squeeze; rock bursts were unknown. In 1931 every stope back in the mine was squeezing and crushing, and rock bursts were frequent.

The use of scrapers in the drifts and subdrifts has reduced the cost of drifting nearly $2 per foot and increased the speed of drifting. The use of scrapers in the stopes has been very satisfactory. With the stope car, the greatest amount two men could shovel, when no picking was done, was about 30 tons per 8-hr. shift. With the scraper, when no picking is done, two men can handle 40 tons per hour. Under the usual conditions in the stopes, when the rock is hand-picked, the use of the scraper does not give such a large increase in efficiency. There is, however, a great advantage over hand loading even when picking is done; there is no track or sollar to be laid; the time of handling rock from stope to raise is negligible; there is no shoveling of fines, a slow operation; even in lean stopes there are frequent rich bunches of copper which cannot be picked, and the scraper moves this rock at a much lower cost than that of hand loading.

All the stopes now being worked are crushing, owing to the weight of the hanging country. Stoping must be conducted with unusual care. It is already evident that loose and "heavy" ground can be much more effectively controlled in the sublevel inclined cut-and-fill method than in the horizontal cut-and-fill system. In the subs, props and legs stand on solid ground; in the horizontal stopes they stand on fill. By adequately supporting the brow or bottom of the inclined back, a certain

amount of cantilever support is obtained upward along the stope back, which when augmented by small props placed in the stope, bottomed on the fill and pitching against the stope back, gives protection to the men working in the stope. The time element in crushing is important; and by taking only comparatively thin slices (3½ to 4 ft. thick) up through the stope, the time of exposure under any one stope back is only a few shifts, and the men are constantly retreating from the stoped-out areas.

Level maintenance expense has been reduced to a minimum.

Suggested Variations of the Sublevel Inclined Cut-and-fill Stoping System

After a study of mining conditions and methods employed in many mining districts in the United States, Canada and Mexico, the authors are of the opinion that the system of sublevel, cut-and-fill stoping developed at the Champion mine could, with certain modifications to meet local conditions, be profitably substituted in some other mines or sections thereof.

The usual cut-and-fill method in which stopes are carried up from level to level in a single lift is a comparatively slow method of stoping. Where, as in the deep mines of the Keeweenaw Peninsula of Michigan, it is necessary or advisable to mine on the retreat, the disadvantages of a slow stoping method are greatly aggravated by the restricted stoping area on each level. In other districts, where the ground is heavy, the use of a retreat system would often be feasible and advantageous provided a rapid stoping method could be devised to suit the conditions.

In ground requiring the use of a cut-and-fill method of stoping, the sublevel inclined cut-and-fill system may offer some or all of the following advantages:

1. High rate of extraction from a block of limited size; of particular advantage when mining on the retreat.

2. In weak ground the time element often has an important bearing upon the extent of failure of ore and wall rocks before ore extraction is completed and the affected area can be abandoned. Rapid extraction of the ore reduces the extent of failure, the cost of artificial support and maintenance, and the hazards of working.

3. For similar reasons the dilution of ore by waste from the walls would be reduced in some instances and often more complete extraction of the ore would result.

4. This system permits of close supervision by a minimum number of bosses, owing to a high degree of concentration of operations, which promotes efficiency and safety.

5. Filling is run into place by gravity with little or no hand work required for spreading and leveling off the fill, an advantage possessed

also by the ordinary rill stoping system. In the latter, however, the stopes are usually long, there is inadequate space in the stope for storage of floor planking during filling, and considerable time and labor are expended in handling plank, timber, drills, gear and drill steel in and out of the stope between the different cycles in the operation. Also, little

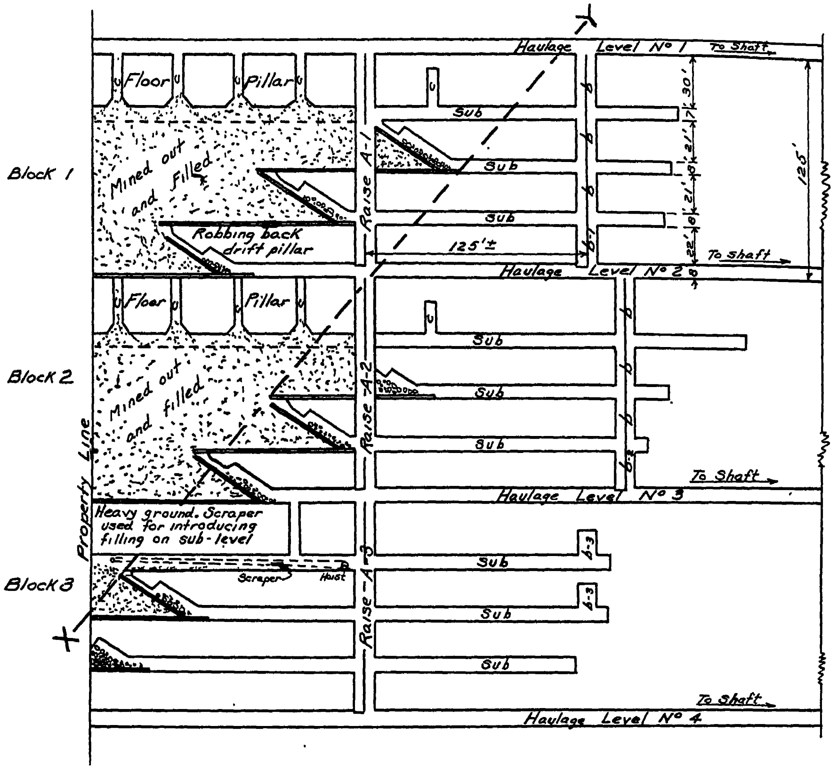

FIG. 3.—FULL RETREAT SYSTEM OF SUBLEVEL CUT-AND-FILL STOPING, USING POWER SCRAPERS ON EACH SUBLEVEL.

Raises:

A, through raises driven between haulage levels, timbered for chute and manway.
b, driven as short raises through pillars between subs for chute and manway.
c, small holes in floor pillars for introducing waste filling.

Order of working: (1) Drive haulage levels; (2) drive raises *A*; (3) drive subs full width of ore from raises *A* to property line, using scrapers; (4) start stoping at property line, retreating as shown, then drive subs on opposite side of raises *A*; (5) put up waste raises *C* as required for filling, connect levels by short raises *b* and timber through; (6) run in filling through raises *C* as required to keep blocks full or scrape fill in as shown in block 3.

opportunity is afforded for keeping blocks of waste out of the ore. In the sublevel system only a relatively small amount of flooring is required and there is ample space in the sublevel back of the toe of the stope for storage of material, tools and supplies. The sublevel system, therefore, possesses the advantages of the usual rill system while its disadvantages are reduced to a minimum.

6. The sublevel system affords ideal conditions for the use of power scrapers with attendant advantages as compared to hand work, since the scraper works over a solid bottom requiring no flooring and the ore is scraped directly to chutes without any delays incident to waiting for cars.

7. All the operations involved are standard, consisting of drifting, taking down backs on the incline, and scraping of the broken ore back to the chute raises.

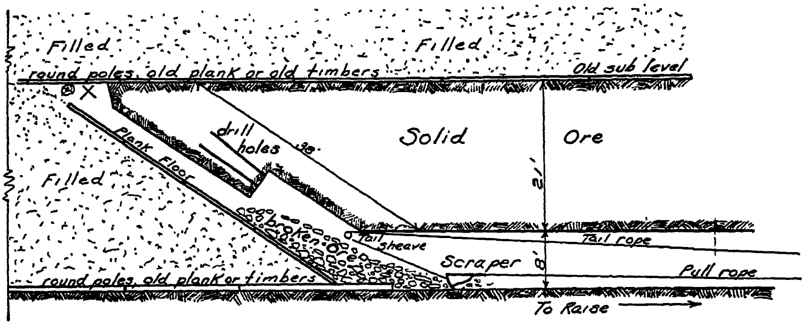

FIG. 4.—SECTION AT FACE OF STOPE.
To run in filling blast out poles at X.

8. Ore broken in taking down backs is often the cheapest ore secured in the mine; power scraping affords a cheap method of handling the broken ore and only the driving of the sublevels would ordinarily be considered expensive. However, by driving large subdrifts and using power scrapers, the cost per ton of ore broken in the drifts is not over

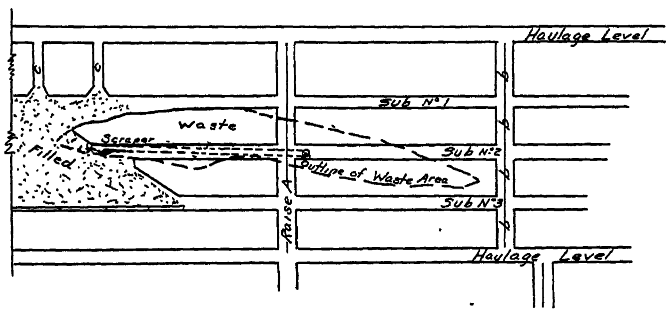

FIG. 5.—METHOD OF MINING AND FILLING AROUND A WASTE AREA.
An alternate method would be to mine sub No. 3 by horizontal cut and fill; scraping filling back from raise *A*.

one-half that for narrow drifting with hand loading. The driving of sublevels serves to thoroughly prospect the ore block before stoping begins, thus permitting better planning, with its attendant advantages.

9. The method commends itself from the standpoint of safety, since the stopes are short, permitting quick retreat of the workmen; the broken ore runs down to the toe of the pile and the shovelers or scraper attend-

ant remain at one place close to the back, which can be inspected easily and made secure.

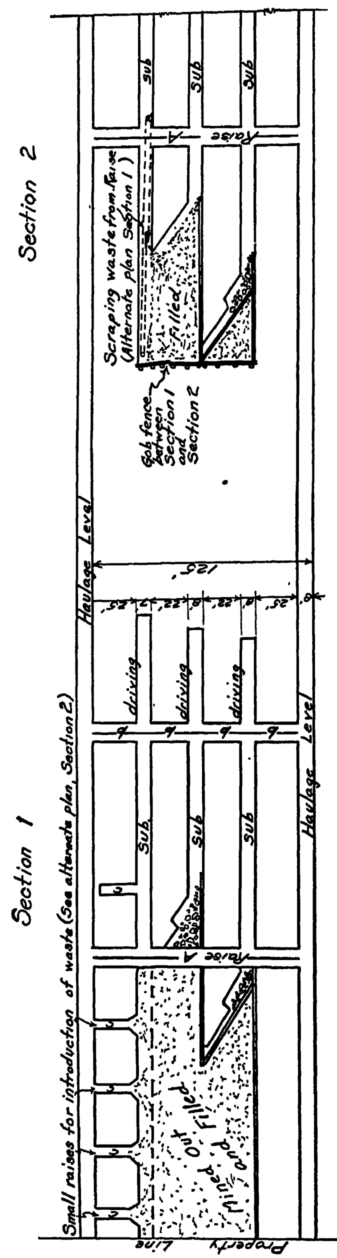

FIG. 6.—SECTIONAL SYSTEM OF SUBLEVEL CUT-AND-FILL STOPING. Raise *A* driven from level to level and timbered for chute and manway. *b*, short raises from sub to sub timbered for chute and manway.

10. By working on the retreat, level maintenance costs are reduced to a minimum, since by the time the level becomes heavy under the stope area the ore will usually all have been extracted and that portion of the level can be abandoned.

Variations of Method

The details of the method developed at the Champion mine were worked out to fit the particular conditions at that mine. The method appears to be flexible, however, and capable of adaptation to other conditions by varying the details of its application.

The following discussion is presented with a view to showing how the method might be varied to suit a set of conditions different from those at the Champion mine. Other possible variations will suggest themselves to the reader.

The assumed conditions are actual ones and the illustrations (Figs. 3, 4, 5 and 6) show how the system might be varied to suit them.

The ore is high grade, is very hard and lies between weak and badly broken walls which in some places peel off in large slabs after exposure for a short time, and in others break out in large blocks to fault planes, or to smaller fractures and slips in the rocks. Mined-out areas must be kept filled with waste rock or sand to prevent subsidence of the hanging wall and damage to shafts, upper workings and surface improvements.

The haulage levels are 125 ft. apart vertically and three subs are driven between levels. A greater interval between haulage levels would be advantageous, but the illustration (Fig. 3) shows a plan of working devised for a particular mine in which the level interval was already established. After driving the haulage levels to the property line or end of the orebody, starting raises *A*-1, *A*-2 and *A*-3 are driven through from level to level and timbered to provide chute and manway compartments. These are the only long raises required, the subsequent timbered raises being driven as short connecting stubs *b*-*b*-*b* between sublevels, thus greatly decreasing their cost as compared to the driving of long raises.

Sublevels are driven in both directions from the starting raise, as shown, those to the left in the illustration being pushed at the expense of those to the right if necessary. Stoping starts as soon as the middle sub reaches the line. In this particular case the capping must be supported against caving and cannot be drawn for filling. Therefore, a floor pillar is left under the haulage level through which small holes *c* are driven as required for the introduction of filling. A floor pillar probably would have to be retained more or less permanently at every fourth or fifth haulage level. In block 1, the drift pillar is shown partly robbed back over the second haulage level, and the floor pillar below that level can now be pulled back.

In some ground the waste holes *c* might weaken the floor pillars too much, in which case the filling could be scraped in from a single waste raise as shown in block 3. In any event the filling for the lower stopes would be drawn from the filled stopes above, where the loss of filling would be made up by waste dumped in from above. Where possible, a preferred method would be to carry the stopes along a continuous diagonal line across a number of levels as line *X*-*Y*, Fig. 3.

Another alternative, which might be more satisfactory in some conditions, would be to mine out the level in the same manner as a sub, holding the walls and fill on stulls or timber sets and scraping the filling material back into the stope. No floor pillar would be left, but this block of ore would be mined from the sub below in its regular sequence.

Fig. 4 shows an enlarged section at the face of the stope. In this particular case the ore is high grade, carrying high metal content in the fines. Therefore a plank floor would be laid on the filling, upon which the ore would be blasted. Also, to prevent contamination of the ore with waste from the fill when stoping up to it from below, it would be necessary to lay a mat of poles, plank or old timber on the bottom of the sublevels before placing the filling. Mining up to this mat would be no different from mining to a mat in top-slice mining as far as safety, complete extraction of the ore, and elimination of dilution by waste are concerned. When the ore has been completely removed from the cut, the sloping plank floor has been removed and a mat has been laid,

filling would be let in by the simple expedient of blasting out the mat as at *X*, Fig. 4. This blasting would be done so as to break the mat timbers into short pieces in narrow orebodies, since long pieces might lodge between the walls and hang up the waste, leaving dangerous empty holes.

The broken ore would be scraped to the ore pass with power scrapers, one to each sublevel. The same scraper would be employed in advancing the subdrift on the opposite side of the raise, concurrently with the stoping operation. Thus by the time the ground is stoped out to the raise, as in block 1, Fig. 3, a new manway and ore pass will be ready for use.

In Fig. 5 is shown a method of mining and filling around a waste area or large horse in the orebody. One compartment of the raise would be used temporarily for supplying waste filling, which would be dragged to the stope by the power scraper. An alternate scheme would be to mine sub No. 3 by ordinary horizontal cut-and-fill, using a scraper for handling the ore from the face to the raise and afterward for dragging the filling from the raise back to the stope.

If sufficient levels cannot be made available to supply the required tonnage by a full retreat system as shown in Fig. 3, a sectional system as illustrated in Fig. 6 could be employed, each section being mined on the retreat as shown. A gob fence could be put in on the left of section 2, so that when mining up to it from section 1, contamination of the ore from the filling material in section 2 would be avoided.

While the illustrations show variations of the sublevel cut-and-fill system for adaptation to a certain set of conditions, the method developed at the Champion mine is believed to be susceptible to other variations as to detail and is flexible enough to permit the adoption of various expedients to take care of special local conditions as they may occur.

Economic Size of Metal-mine Airways*

By G. E. McElroy,† Pittsburgh, Pa.

(New York Meeting, February, 1932)

Changes in existing airway and fan-installation conditions offer the most common opportunities for effecting economical operation of mine-ventilating systems, but the largest possibilities for securing this result lie in the original design of the main airways for efficient service. Formulas, and charts for their graphical solution, are presented herewith for the determination of the elements of airways design when the conditions of service are known or may be roughly approximated. Although the analyses are made with particular reference to metal-mine airways, there is no essential difference between main airways in metal and coal mines, and the methods presented are applicable to both.

Little attention is paid to designing airways for maximum economy in the ventilation of metal mines. Most of the openings used as main airways are originally designed and used for transportation and other operating purposes. Even where openings are made primarily for ventilation, the possibility of their future use for other operating purposes, or even custom alone, usually results in the selection of a size and shape similar to those of existing openings. Natural conditions, such as heavy ground, often seriously limit the size of opening that can be maintained without excessive expense; and various operating factors, such as velocity limitations on traveling roads, often dictate size requirements in excess of those required for economy in ventilation only. In metal mines the main airways are, as a rule, too small rather than too large for economic service, with the general result that the quantities in circulation are often less than the mine's proper requirements over long periods, even though excessive amounts of power are applied in an effort to remedy conditions.

Increasing the area of an airway increases the cost for excavation and lining practically proportionately but at the same time affects a very rapid decrease in power requirements. The balancing of these two factors of the expense to give a minimum total yearly operating cost for ventilation is mainly a matter of proper original design of the main airways of the mine, since the large volumes handled through these parts of the system involve large power costs and thus present opportunities for effecting large savings through proper design. Many of the factors

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required for mathematical analysis are not known accurately in most cases and some may have very indefinite values, yet the magnitude of the savings possible through economic design is such that extensive mathematical investigation is warranted under a much greater range of conditions than it is now, or probably ever will be, applied. And a mathematical analysis can yield reasonable accuracy even though many of the factors involved can be assigned only the most approximate values.

Summary and Conclusions

Total yearly costs for passing air through mine airways are primarily a question of unit costs of construction and operation, quantity of flow and size of airway. All except the latter are fixed by the conditions obtaining, and design centers on making the size of the airway such that yearly costs are a minimum for the existing conditions. The mathematical analyses made herewith lead to the following conclusions:

1. The area for maximum economy is determined both by the absolute value of a dozen or more separate factors and by the relative value of the two groups into which they naturally divide.

2. The relation which exists between capital-cost factors and power-cost factors largely determines the area required for economy, which might vary as much as 4 to 1 for probable changes met in practice.

3. Of the separate factors involved, quantity of flow is most important in that it affects the economic area at a rate three times that of all of the other separate variables. Range of variation in probable values of the other separate factors determines their importance as affecting economic area. Character of wall surfaces and ratio of unit cost of lining to unit cost of excavation are the two other separate factors of major importance. Unit cost of excavation, unit cost of power, shape, lining thickness, service life, interest rates, and mechanical efficiency of fans have but minor effects as to economic area, decreasing in importance in the order given.

The results of many calculations for assumed factor values lead to the following deductions:

1. Total costs per unit quantity of flow may vary as much as 20 to 1 for probable variations in the values of the factors involved, particularly unit costs. Mine-ventilation costs, expressed on a similar basis, cannot help varying accordingly and therefore do not furnish, by themselves, any direct basis for judging the economics of a ventilating system.

2. Under conditions of comparable unit costs, a probable range of variations in total costs per unit quantity of flow, as affected by design, of about 4 to 1 seems possible. Unlined airways, where feasible, have a large advantage over lined airways, particularly in the case of those with the smoother walls. The comparative advantages of different types of lined airways depend to a large extent on the ratio of unit cost of lining to unit cost of excavation. Under ordinary conditions, it seems probable

that they would rank in the following order of decreasing merit as airways: smooth lining inside open-frame timber, solid crib timber, concrete, and open-frame timber.

3. Shape of airway has but a minor effect on total costs and becomes important only when the other major factors of design are fixed. On a constant-factor basis, circular airways have a slight advantage over single-compartment rectangular airways; and the latter have a similar slight advantage over multicompartment airways.

4. Where operating conditions limit the size of airways and require larger or smaller areas than the area for maximum economy of air flow, the approximate cost of the limiting condition may be determined by the methods of mathematical analysis for economic area presented herewith.

Mathematical Analysis

Power requirements are determined by the quantity of flow and the pressure required to cause the flow. The latter must balance the pressure losses, which are composed of friction pressure losses due to wall surfaces, and velocity pressure losses due to deflections from a straight line and changes in area. Main airways are usually straight and of uniform area of cross-section. Velocity pressure losses may be involved, but usually they can be ignored as constituting but a minor component of the power requirements. Mathematical analyses can therefore be based entirely on friction pressure losses or, at the most, on friction factors that include an allowance for all but local velocity pressure losses.

The yearly capital charge is sometimes obtained by distributing the total cost of construction uniformly over the period of service life, but this procedure results in very low yearly capital charges with no provision for proper return on the money invested in the airway. The total yearly capital charge against the airway should be a fixed percentage of the total first cost of excavation and lining. This fixed percentage is that determined by the sum of the yearly interest requirements on the money involved, as fixed by what the money is worth to the operator, and the yearly amount that, placed at interest, would return, or amortize, the total amount of the investment at the end of a period of years equivalent to the estimated service life. The latter can be found in sinking-fund and annuity tables and expressed as a percentage of the first cost. The service life of a metal-mine airway is usually a matter of rough estimate only and, since the amortization percentage is largely a function of service life rather than interest rate and but one of the minor factors in the mathematical analysis, the few values of Table 1 should suffice for ordinary requirements.

The sum of the interest and amortization percentage, expressed as a rate, $\frac{\text{percentage}}{100}$, is termed the capital return rate, and determines

the yearly amount that should be charged against the airway as a capital charge.

TABLE 1.—*Annual Payment, in Per Cent of a Given Sum, Which, if Set Aside at the End of Each Year, Will Amount at r Per Cent Interest, Compounded Annually, to the Given Sum at the End of a Period of Years*

	Service life in years									
	2	3	4	5	7	10	15	20	30	50
	Annual payment, per cent									
At 3 per cent interest..	49.3	32.4	23.9	18.8	13.1	8.7	5.4	3.7	2.1	0.9
At 6 per cent interest..	48.5	31.4	22.9	17.7	11.9	7.6	4.3	2.7	1.3	0.3

Maintenance of airways is usually independent of design to a large extent. Where it is a factor in design, it is best handled by a proportional increase in the capital return rate to cover maintenance charges as well as the purely capital charges of interest and amortization of principal.

SIMPLE SOLUTION FOR CONSTANT CONDITIONS

With all factors and conditions constant except area of cross-section, a simple solution may be made for the area for maximum economy by calculating separately the total yearly cost, under the best assumptions that can be made, for a number of assumed areas and plotting the results against area. The low point of the resulting total-cost curve determines the area that would be most economical under the assumed conditions. It will be noted that the curve is quite flat in the vicinity of the low point, showing that a considerable variation in area is permissible without materially increasing the total yearly cost. Weeks[1] solves an example by this method and cites a calculated saving of $4778 a year, with a certain set of conditions and unit costs, by the use of a 10-ft. rather than an 8-ft. diameter unlined shaft 1000 ft. long for handling 100,000 cu. ft. per minute. Hall[2] gives a table of total yearly costs, for a fixed set of conditions and unit costs, for passing quantities varying from 100,000 to 600,000 cu. ft. per minute through 600 ft. of shafts 6 to 30 ft. square, lined with concrete 1 ft. thick; and cites a calculated accumulative saving of $44,786,000 in 50 years as a possible result of using a 20 by 20-ft. shaft rather than a 10 by 10-ft. shaft for a flow of 500,000 cu. ft. per minute through but 600 ft. of shaft.

[1] W. S. Weeks: Ventilation of Mines, 78. New York, 1926, McGraw-Hill Book Co.
[2] M. H. Hall: How to Find Economical Size for Airshaft. *Coal Age* (1922) **22**, 361.

With the calculations made for one assumed area, A_o, the area for maximum economy, A_m, may be found directly from the relation:

$$A_m = \frac{A_o}{0.77\left(\dfrac{C_{co}}{C_{po}}\right)^{2/7}}$$

where C_{co} and C_{po} are the yearly capital charge and yearly power cost based on the assumed area, A_o.

This formula, whose derivation, as shown in Appendix 1, follows that used by Busey and Carrier,[3] is based on friction pressure losses only and considers that for lined airways the thickness of the lining varies directly with the diameter for circular airways, or with the longer side of rectangular airways. The derivation of a similar formula for velocity pressure losses only is shown in Appendix 2, in which the constant is approximately the same, but the fractional exponent is one-third. The difference in formulas is small and ordinarily velocity pressure losses are not significant in the design of main airways, so that the above formula only is required for adequate, approximate solutions. The equation may also be put in the form:

$$\left(\frac{A_o}{A_m}\right)^{7/2} = 0.4\frac{C_{co}}{C_{po}}; \text{ so when } A_o = A_m, \left(\frac{A_o}{A_m}\right)^{7/2} = 1 \text{ and } C_p = 0.4C_c.$$

That is, when the area is that for maximum economy, the power cost is 40 per cent of the yearly capital cost. Assuming, as an average figure for metal-mine airways, that the yearly capital charge is 20 per cent of the original cost of construction, then the yearly power cost should be about 8 per cent of the total cost of excavation and lining.

Investigation of Elements of Design

General Formula

A wise selection can rarely be made through consideration of one set of conditions only; it is generally desirable to investigate the effect of a number of probable variations in conditions and unit costs on design and final cost, and effect a practical compromise. Although the formula given can be used for this purpose, a detailed analysis of the relative effect of the dozen or more variables involved in any practical solution requires a more general type of formula, which takes the form:

$$A_m = \left(\frac{10^9 KFc_p}{13{,}200EI(c_eG + c_sJ)}\right)^{2/7}\left(\frac{Q}{1000}\right)^{6/7}$$

[3] F. L. Busey and W. H. Carrier: The Design of Indirect Heating Systems with Respect to Maximum Economy of Operation. *Jnl.* Amer. Soc. Heat. and Vent. Engrs. (1913) **19**.141–163.

where A_m is free area for air flow for maximum economy, in square feet,
K is friction factor for foot-pound-minute units used in mine ventilation,
F is shape factor and equals $\frac{\text{perimeter in feet}}{\text{square root of area in square feet}}$,
c_p is unit cost of power per horsepower-year, in dollars,
E is over-all mechanical efficiency ratio of the power-using source,
I is rate of capital return, or interest per cent plus amortization per cent divided by 100,
c_e is unit cost of excavation per cubic foot, in dollars,
G is ratio of total area of excavation to free area of flow,
c_s is unit cost of lining *per cubic foot,*
J is ratio of area of excavation required for lining to free area for flow, and
Q is quantity of flow in cubic feet per minute.

The derivation of this formula, as shown in Appendix 3, follows that used by Richardson.[4] It considers friction pressure losses only as being the predominating cause of power cost; and is based on the assumption that the thickness of lining varies directly with the diameter, or longer side, as it should do theoretically, even though in practice a constant thickness, based on ground conditions, is the prevailing method of determining thickness of lining. However, a reasonably close estimate can be made for thickness of lining in terms of ratio of diameter or longer side for most conditions and a trial solution can be made for area and dimensions; if there is then any large discrepancy, a revised thickness ratio can be used in a new calculation and the process repeated until the resultant thickness agrees with the estimate used. As a rule, not more than two solutions are required to give reasonably close agreement.

The formula also assumes that all of the power required for the flow is mechanically applied and paid for. Actually, some of the power is always supplied gratis by natural draft. For shallow mines, even though the natural draft pressure may at times be against the fan pressure, the average for the year is with the fan pressure, but the natural draft power in shallow mines is usually such a small percentage of the total power that it can be neglected in computations for economical area. In deep metal mines the natural draft pressure is usually with the fan at all times, but it varies in intensity seasonally and often contributes power gratis equivalent to one-fourth to one-third the total power requirements. Total power requirements vary as the cube of the quantity, whereas the natural draft power supplied varies approximately directly as the quantity and is such a variable factor that it is difficult to include it in

[4] A. S. Richardson: Economic Design of Mine Airways. *Trans.* A.I.M.E. (1926) **74,** 342–351.

a mathematical solution. It can be taken into account approximately by reducing the unit cost of power proportional to the estimated ratio of paid-for power to total power, and finding a final solution through successive trial solutions. As in the case of thickness ratio, not more than two solutions would generally be required for close agreement.

Shape, Excavation and Lining Factors: F, G and J

The shape factor F is introduced to obviate the necessity of using the double variable p/A, or perimeter divided by area, and can be found from the relation $F = \frac{\text{perimeter}}{\text{square root of area}}$. Values of F for a number of common shapes of airways are given in Table 2.

The excavation factor G is 1 for unlined airways, and for lined airways has a value determined primarily by the thickness of the lining and secondarily by the shape and number of compartments. Without allowance for overbreak, and assuming, in the case of compartmented airways, that the thickness of the dividers is the same as the wall lining—an assumption that is rarely more than a rough approximation but sufficiently accurate for our purpose—the general expression for G, as derived in Appendix 4, is:

$$G = \frac{\text{Total area of excavation}}{\text{Free area for flow}} = \frac{N + RM + NRM}{N}(1 + 2R)$$

where N is the number of similar compartments of equal size in line,
M is the ratio of longer to shorter side of rectangle, and
R is the ratio of thickness of lining to longer side (or diameter).

For a single compartment, $N = 1$, and

$$G = (1 + 2RM)\,(1 + 2R).$$

For a square, regular polygon, or circle, $M = 1$ also, and

$$G = (1 + 2R)^2.$$

For a single compartment with three sides lined, with a shorter side unlined

$$G = (1 + R)\,(1 + 2RM).$$

But with a longer side unlined, it is

$$G = (1 + 2R)\,(1 + RM).$$

Values of G for common shapes and thickness ratios may be interpolated from the values listed in Table 2, or may be graphically computed by means of Fig. 1. Values for particular designs may be determined easily from dimension data or through similar formulas derived by expressing both total area and free area algebraically in terms of the separate dimensions.

For flow through a multiple airway system, common in coal-mine ventilation, solution should be made for each airway for the proportional part of the total flow that it is to handle. If there are N airways of equal size, each would handle $1/N$ of the total flow. On account of the relation that exists between economic area and quantity of flow, the total area and cost for multiple airways would be larger than for a single airway accommodating the total flow if all other factors remained constant, which, of course, is not usually the case in practice.

TABLE 2.—*Shape Factors, F, for Common Designs of Metal-mine Airways and Excavation Factors, G, for Various Lining Thickness Ratios, R*[a]

Shape and design of airway	Side Ratio, m	Number of Compartments, N	Shape Factor, F	Excavation Factor, G			
				$R = 0.10$	$R = 0.15$	$R = 0.20$	$R = 0.25$
Circular			3.55	1.440	1.690	1.960	2.250
Octagonal			3.63	1.440	1.690	1.960	2.250
Rectangular, three sides lined, shorter side not lined	1.00	1	4.00	1.320	1.495	1.680	1.875
	1.25	1	4.03	1.375	1.581	1.800	2.031
	1.50	1	4.08	1.430	1.668	1.920	2.188
	2.00	1	4.24	1.540	1.840	1.160	2.500
Rectangular, three sides lined, longer side not lined	1.00	1	4.00	1.320	1.495	1.680	1.875
	1.25	1	4.03	1.350	1.544	1.750	1.968
	1.50	1	4.08	1.380	1.593	1.820	2.063
	2.00	1	4.24	1.440	1.690	1.960	2.250
Rectangular, four sides lined	1.00	1	4.00	1.440	1.690	1.960	2.250
	1.25	1	4.03	1.500	1.788	2.100	2.438
	1.50	1	4.08	1.560	1.885	2.240	2.625
	2.00	1	4.24	1.680	2.080	2.520	3.000
Similar rectangular compartments in line	1.00	2	5.66	1.380	1.593	1.820	2.063
	1.25	2	5.69	1.425	1.665	1.925	2.203
	1.50	2	5.77	1.470	1.738	2.030	2.343
	2.00	2	6.00	1.560	1.885	2.240	2.625
	1.00	3	6.93	1.360	1.560	1.773	2.000
	1.25	3	6.97	1.400	1.625	1.867	2.125
	1.50	3	7.07	1.440	1.690	1.960	2.250
	2.00	3	7.34	1.520	1.820	2.147	2.500
	1.00	4	8.00	1.350	1.543	1.750	1.968
	1.25	4	8.05	1.388	1.604	1.838	2.085
	1.50	4	8.16	1.425	1.667	1.925	2.202
	2.00	4	8.48	1.500	1.788	2.100	2.437

[a] G is ratio of total excavation to outside of lining only, with no allowance for overbreak, to excavation required for free area only, with dividers assumed to be of the same thickness as the wall lining. R is ratio of thickness of lining to diameter, or longer side of rectangle, measured inside the lining.

The values of G, as derived through the given formulas, table, and chart, do not include overbreak, but, if desired, an allowance for over-

break may be included in G by increasing the nominal value by the estimated percentage overbreak; for instance, for 10 per cent overbreak, G would be 1.10 times the nominal value as based on lining thickness only, or $G(1 + y)$ where y is the ratio of overbreak in terms of total nominal area.

Where the overbreak affects both excavation and lining, the value of the lining factor J is $G - 1$; but in the case of timbered-lined airways this relation holds only if the unit lining cost is based not on the actual space occupied by the lining but on the space occupied by both lining and overbreak. If unit costs are based on lining only, $J = G - 1$ only when G is the nominal value of the excavation factor without allowance for overbreak.

Combined Excavation and Lining Factor, X

In order to facilitate comparison of effects and graphical methods of solution, it is desirable to eliminate the plus sign in the first member of the general equation by substituting c_eX for $(c_eG + c_sJ)$, where X is a combined factor for both excavation and lining. The general equation then becomes:

$$A_m = \left(\frac{10^9 KFc_p}{13{,}200EIc_eX}\right)^{2/3} \left(\frac{Q}{1000}\right)^{6/3}$$

If we let

$$B = \left(\frac{10^9 KFc_p}{13{,}200EIc_eX}\right) \text{ and } Z = B^{2/3}$$

$$A_m = Z\left(\frac{Q}{1000}\right)^{6/3}.$$

For an unlined airway, G is 1, J is 0, and the value of X is 1.

For a lined airway without allowance for overbreak, or with the overbreak affecting both the excavation and lining, as in a concrete-lined airway, $J = G - 1$ whether G is the nominal value for the excavation factor, as determined by the nominal dimensions outside the lining, or the nominal value increased a proportionate amount for overbreak, and

$$X = \left(\frac{c_s}{c_e} + 1\right)G - \frac{c_s}{c_e}$$

For a timbered-lined airway, with allowance for overbreak affecting the excavation only, $J = G - 1$, where G is the nominal value of the excavation factor and, on this basis, the actual value of the excavation factor is $(1 + y)G$, where $y = \frac{\text{percentage overbreak}}{100}$, and

$$X = \left(\frac{c_s}{c_e} + 1 + y\right)G - \frac{c_s}{c_e}.$$

Since y does not affect the value of X more than about 3 per cent for the maximum range, encountered in practice, of the other factors involved, it is permissible to neglect it and use the first formula given for X as a standard formula, using the nominal value of G, without allowance for overbreak, in all cases except where the quantity of lining is affected by overbreak, for which the nominal value of G should be increased in proportion to the estimated overbreak.

Effect of Variation in Factors on Economic Area

Costs are determined primarily by area and unit costs. Total costs will of course vary with unit costs, but these are not considered as subject to manipulation and the factor that determines economic design is the area for free flow.

The greatest rate of change in A_m is produced by variations in quantity of flow, and is three times the rate of change affected by any of the other factors as represented in the formula, for which reason the quantity term has been segregated from the rest. Some idea of the rate of change in A_m effected by changes in quantity may be gathered from a few examples:

For 5000 cu. ft. per minute, $A_m = 3.97\ Z$, or $0.79\ Z$ per 1000 cu. ft. per minute.

For 100,000 cu. ft. per minute, $A_m = 51.8\ Z$, or $0.52\ Z$ per 1000 cu. ft. per minute.

For 500,000 cu. ft. per minute, $A_m = 205.8\ Z$, or $0.41\ Z$ per 1000 cu. ft. per minute.

Since the rate at which quantity affects the economic area is three times that of any other one variable, and since the possible range of values for quantity is also as large as or larger than for any other factor, quantity of flow is the major single variable influencing economic area.

With quantity constant, the economic area is determined by the value of B for the particular design chosen. B is a maximum when all of the factors above the line have maximum values and all those below the line minimum values, and vice versa. An extremely large range in B values is therefore possible, and although the rate of change thus effected in Z and A_m values is less than for quantity changes, a very large effect on A_m is possible. Since the factors above the line are all factors affecting power costs and since those below the line, with the exception of efficiency, E, which has but a limited range, are all factors affecting the capital charge, it follows that the major changes in this term are those due to the comparative relation of power factors to capital factors; and the economic area is largely a matter of the relation that exists between these two groups of factors.

Each of the separate variables affects the economical area at the same rate, and their practical effect in causing a change in area is thus

a matter of the relative range of values that may be encountered in practice. The ratio of minimum to maximum values ordinarily considered for the separate factors is rarely greater than 2 to 1 for F, E and I, or 4 to 1 for c_p and c_e, or 15 to 1 for K and X. The effect on economic area is proportional and is therefore relatively unimportant for F, E and I; c_p and c_e are of slight importance; but the really important factors are K and X—that is, the character of the wall surfaces and the lining. In evaluating X, it is found to depend largely on the ratio of unit cost of lining to unit cost of excavation, or c_s/c_e, which has a practical minimum-maximum ratio of about 6 to 1, and to a much less extent on the ratio of total area of excavation to free area G, where the range usually is not over 3 to 1. Next to quantity, therefore, the important separate factors affecting economical area are the character of the wall surface and the ratio of unit lining cost to unit excavation cost.

Effect of Design on Economic Area and Total Yearly Costs

In judging of the economic value of different types of openings for use as airways, the total yearly operating cost—power cost plus capital charge—is a proper criterion, provided the values assigned the different variables agree with average practice. The selection of a representative set of values is largely a matter of personal judgment and experience and is easily subject to controversy. However, in order to form an approximate idea of the relation that exists, the author has assumed sets of roughly corresponding values for the variables and has calculated three groups of area and cost data, for 100,000 cu. ft. per minute through 1000 ft. of airway, which are given in Table 3. In the first group, various types of airways are examined for comparable assumed values of the factors. They show that, where feasible, unlined airways cost about one-half as much as lined airways; that circular airways have a small advantage over rectangular airways; and that single compartments have only a slight advantage over multicompartment airways. In determining the relative value of several types of lined airways with the unit cost of excavation constant, the relative unit costs assumed for lining practically control the results obtained. The figures hold only, therefore, for the relative values shown in the table, and, as assumed, indicate a small advantage for thin-board lining inside open-timber framing, and for solid cribbing over concrete lining; and a slight advantage for the latter over open-timber lining. However, small changes in the assumed unit cost of lining for each type would change the results slightly, and the differences are so small that, in any particular case, they might easily be offset by operating and maintenance advantages or disadvantages.

Effect of Relation between Cost Factors

A much larger variation in possible total yearly costs is shown in the second and third groups of calculated data in Table 3, where the effect of

TABLE 3.—*Design Factor, Z, for Common Types of Metal-mine Airways for Selected Values of the Controlling Factors, Economic Size for Flow of 100,000 Cubic Feet per Minute, and Yearly Costs per 1000 Linear Feet of Airway*

Type of Airway and Lining (Ratio of Longer to Shorter Side of Rectangles 1.25 in All Cases) and Relative Value of Factors	Selected Factors: Friction Factor	Selected Factors: Ratio of Over-all Mechanical Efficiency	Selected Factors: Lining Thickness, Ratio	Selected Factors: Ratio of Capital Return	Unit Cost: Power, per Hp.-yr.	Unit Cost: Excavation, per Cu. Ft.	Unit Cost: Lining, per Cu. Ft.	Ratio of Excavation Area to Free Area	Combined Excavation and Lining Factor	Design Factor	For Flow of 100,000 Cu. Ft. per Minute: Free Area, Sq. Ft.	Dimensions, Ft.	Velocity, Ft. per Min.	Annual Expense per 1000 Ft. of Airway: Capital Charge: Excavation	Capital Charge: Lining	Capital Charge: Total	Power Cost	Total
Nomenclature	K	E	R	I	c_p	c_e	c_s	G	X	Z	A_m	O MO	V_m	C_e	C_s	C_c	C_p	C
Average value of factors:																		
Unlined circular, in sedimentary rock	$0.0^{8}6$	0.60		0.15	$100	$0.50		1.00	1.00	1.440	74.6	9.75 dia.	1,341	$ 5,595		$ 5,595	$ 2,238	$ 7,833
Unlined rectangular, in sedimentary rock	$0.0^{8}6$	0.60		0.15	100	0.50		1.00	1.00	1.493	77.3	7.87 × 9.83	1,293	5,800		5,800	2,320	8,120
Unlined circular, in igneous rock	$0.0^{7}15$	0.60		0.15	100	0.50		1.00	1.00	1.871	96.9	11.11 dia.	1,032	7,267		7,267	2,907	10,174
Unlined rectangular, in igneous rock	$0.0^{7}15$	0.60		0.15	100	0.50		1.00	1.00	1.940	100.5	8.97 × 11.21	995	7,535		7,535	3,014	10,549
Rectangular, open-timber, smooth-lined	$0.0^{8}2$	0.60	0.20	0.15	100	0.50	$0.60	2.10	3.42	0.768	39.8	5.64 × 7.05	2,515	6,262	$ 3,936	10,198	4,079	14,277
Rectangular, solid timber crib lining	$0.0^{8}2$	0.60	0.20	0.15	100	0.50	0.80	2.10	3.86	0.742	38.4	5.54 × 6.93	2,604	6,049	5,070	11,119	4,447	15,566
Circular, concrete lining	$0.0^{8}2$	0.60	0.20	0.15	100	0.50	1.50	1.96	4.84	0.670	34.7	6.65 dia.	2,881	5,103	7,498	12,601	5,041	17,642
Rectangular, concrete lining	$0.0^{8}2$	0.60	0.20	0.15	100	0.50	1.50	2.10	5.40	0.674	34.9	5.28 × 6.61	2,866	5,495	8,636	14,131	5,653	19,784
Three rectangular compartments, concrete lining	$0.0^{8}2$	0.60	0.20	0.15	100	0.50	1.50	1.87	4.48	0.831	43.1	3@3.4 × 4.2	2,322	6,039	8,429	14,468	5,787	20,255
Rectangular, open-timber lining	$0.0^{8}9$	0.60	0.20	0.15	100	0.50	0.50	2.10	3.20	1.203	62.3	7.06 × 8.82	1,606	9,808	5,137	14,945	5,978	20,923
Three rectangular compartments open-timber lining	$0.0^{8}9$	0.60	0.20	0.15	100	0.50	0.50	1.87	2.74	1.470	76.2	3@4.5 × 5.6	1,313	10,681	4,969	15,650	6,260	21,910

TABLE 3.—(*Continued*)

Type of Airway and Lining (Ratio of Longer to Shorter Side of Rectangles 1.25 in All Cases) and Relative Value of Factors	Selected Factors: Friction Factor	Selected Factors: Ratio of Over-all Mechanical Efficiency	Selected Factors: Lining Thickness, Ratio	Selected Factors: Ratio of Capital Return	Selected Factors: Unit Cost: Power, per Hp.-yr.	Selected Factors: Unit Cost: Excavation, per Cu. Ft.	Selected Factors: Unit Cost: Lining, per Cu. Ft.	Ratio of Excavation Area to Free Area	Combined Excavation and Lining Factor	Design Factor	For Flow of 100,000 Cu. Ft. per Minute: Free Area, Sq. Ft.	Dimensions, Ft.	Velocity, Ft. per Min.	Annual Expense per 1000 Ft. of Airway: Capital Charge: Excavation	Capital Charge: Lining	Capital Charge: Total	Power Cost	Total
Nomenclature	K	E	R	I	c_p	c_e	c_s	G	X	Z	A_m	O MO	V_m	C_e	C_s	C_c	C_p	C
Unlined rectangular airway in igneous rock:																		
Values of factors selected to give:																		
Low costs	$0.0^{8}9$	0.65		0.10	50	0.25		1.00	1.00	1.840	95.3	8.74 × 10.92	1,050	$ 2,382		$ 2,382	$ 953	$ 3,335
Large area	$0.0^{7}21$	0.55		0.10	150	0.25		1.00	1.00	3.365	174.3	11.81 × 14.76	574	4,357		4,357	1,743	6,100
Average costs and area	$0.0^{7}15$	0.60		0.15	100	0.50		1.00	1.00	1.940	100.5	8.97 × 11.21	995	7,535		7,535	3,014	10,549
Small area	$0.0^{8}9$	0.65		0.25	50	0.75		1.00	1.00	1.035	53.6	6.55 × 8.19	1,867	10,045		10,045	4,018	14,063
High costs	$0.0^{7}21$	0.55		0.25	150	0.75		1.00	1.00	1.892	98.0	8.85 × 11.07	1,021	18,373		18,373	7,349	25,722
Three rectangular compartments with open-timber lining:																		
Values of factors selected to give:												three compartments each						
Low costs	$0.0^{8}8$	0.65	0.15	0.10	50	0.25	0.25	1.63	2.25	1.648	85.4	4.77 × 5.97	1,171	3,479	$ 1,345	4,824	1,930	6,754
Large area	$0.0^{7}10$	0.55	0.15	0.10	150	0.25	0.25	1.63	2.25	2.522	130.7	5.90 × 7.38	765	5,326	2,059	7,385	2,954	10,339
Average costs and area	$0.0^{8}9$	0.60	0.20	0.15	100	0.50	0.50	1.87	2.74	1.470	76.2	4.51 × 5.63	1,313	10,681	4,969	15,650	6,268	21,910
Small area	$0.0^{8}8$	0.65	0.25	0.25	50	0.75	0.75	2.13	3.26	0.835	43.2	3.40 × 4.24	2,313	17,269	9,162	26,431	10,573	37,004
High costs	$0.0^{7}10$	0.55	0.25	0.25	150	0.75	0.75	2.13	3.26	1.277	66.2	4.20 × 5.25	1,511	26,424	14,018	40,442	16,177	56,619

varying relations between the factors that influence power cost and those that affect capital charges is shown for two widely different types of airways under five different combinations of power and capital cost factors. Here the personal estimate of maximum, minimum, and average values of factors affects only the range of the results, and the selected values are well authenticated in the literature. Low power-cost factors combined with low capital-cost factors give low total yearly costs; medium values in each set give medium total yearly costs; and high values in each set give high total yearly costs. Yet the area for maximum economy changes but little when the two sets of factors occupy such balanced positions. The greatest difference in area results when the factors are not balanced. Thus high power-cost factors combined with low capital-cost factors require maximum areas and low power-cost factors with high capital-cost factors require minimum areas.

Units costs for ventilation, although they will necessarily include many other items, cannot help varying as between one mine and another, and therefore do not furnish, by themselves, any direct basis for judging the degree of economical planning and operation of the ventilating system.

Economical Velocities

The foregoing analysis could also have been made on the basis of velocity, rather than area, since velocity varies inversely with the area for constant quantity. Velocities for maximum economy are also shown in Table 3. They are subject to the same variations as the areas for maximum economy, but in the reverse direction. The economical velocity is therefore also dependent on the particular conditions obtaining and cannot be judged offhand from a small set of sample values. However, both the area and velocity figures in the first group of Table 3 can be used as a rough approximation for general practice, provided that there is a semblance of a balanced relation between power-cost factor and capital-cost factors. In practice, this is not necessarily the case, although it is of quite common occurrence. At the small mine, both sets of cost factors are likely to be high; at the mine of moderate size, medium; and at the large mine, relatively low.

Operating conditions may impose certain limits on velocities of flow, and thus on areas, requiring higher or lower velocities than the economical velocities and larger or smaller areas than the economical areas. The additional cost of complying with the restricting condition may be given a dollars and cents interpretation through the mathematical analyses herewith presented.

Calculation of Economic Area of Airway and Total Yearly Costs

Calculations for economic areas and costs are tedious at best and, since no great degree of accuracy is essential, or possible, on account of

the approximate nature of many of the assumptions that must be made, four charts are presented herewith that permit rapid graphical methods of computation. The calculations are thus divided into four steps: Charts for the first three follow the general formula for economic area and facilitate the determination of X, Z and A_m in progressing order; the fourth chart facilitates the determination of costs and is based on the cost relation developed for economic area—that the yearly power cost is then 40 per cent of the yearly capital charge.

Chart for G and X

Values for G and X are required only for lined airways. Since both are 1 for unlined airways, this step is not required for such types. A table of values for G, the excavation factor, has been presented in Table 2, but a graphic chart is a more convenient method of determining values of X. A chart for determining G and X graphically for common types of lined airways is presented as Fig. 1. The method of solution is indicated, by the dotted line example, for a three-compartment shaft: the intersection of the lining thickness ratio, R, with the ratio of longer to shorter side, M, determines the value of G (allowance for overbreak, made for concrete lining only, should be made at this point by a proportional increase in G); and the intersection of G with the cost ratio, c_s/c_e, determines the value of X. The example shows that when $R = 0.15$, $M = 1.25$ and $N = 3$, $G = 1.625$ and, with $c_s/c_e = 1.0$, $X = 2.25$.

Chart for Z

With the X factor known, the next step is the determination of Z, which may be computed graphically by means of the proportionality chart presented as Fig. 2. In preparing this chart, basic values were selected for the seven variables that would be close to mean values met in practice and still give a Z value of 1.0. The final compromise resulted in the use of $X = 2$ rather than the preferable value of $X = 1$. For convenience in preparing and using the chart, unit power costs are in cents per kilowatt-hour (power cost rate times ratio of operating time for discontinuous operation) and unit costs of construction are in dollars per cubic yard. The chart is based on the fact that the base value of Z will be increased or decreased by the 2/7th power of the ratios of actual values of all of the factors to their selected basic value. The basic values of the seven factors involved are shown by heavy horizontal lines with a flag on the end to indicate the proper direction of diagonal movement to follow to intersection with actual values.

The method of using the chart to determine the value of Z corresponding to the economic area for a particular set of values for the seven factors is shown by the dotted line example. The chart is entered at the $Z = 1.0$

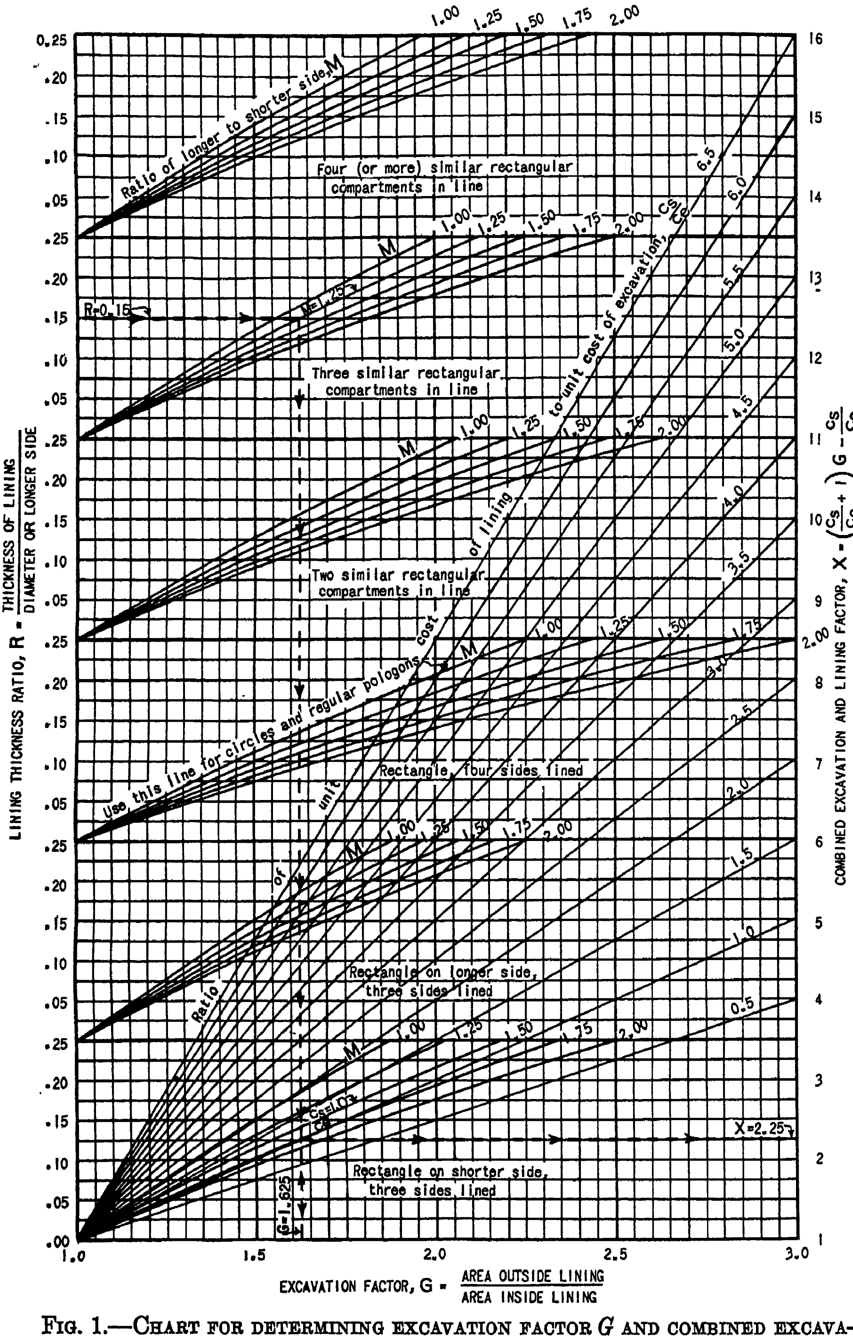

FIG. 1.—CHART FOR DETERMINING EXCAVATION FACTOR G AND COMBINED EXCAVATION AND LINING FACTOR X, FOR USE IN FORMULAS AND CHARTS FOR DETERMINING ECONOMICAL SIZE OF OPENINGS USED AS MINE AIRWAYS.

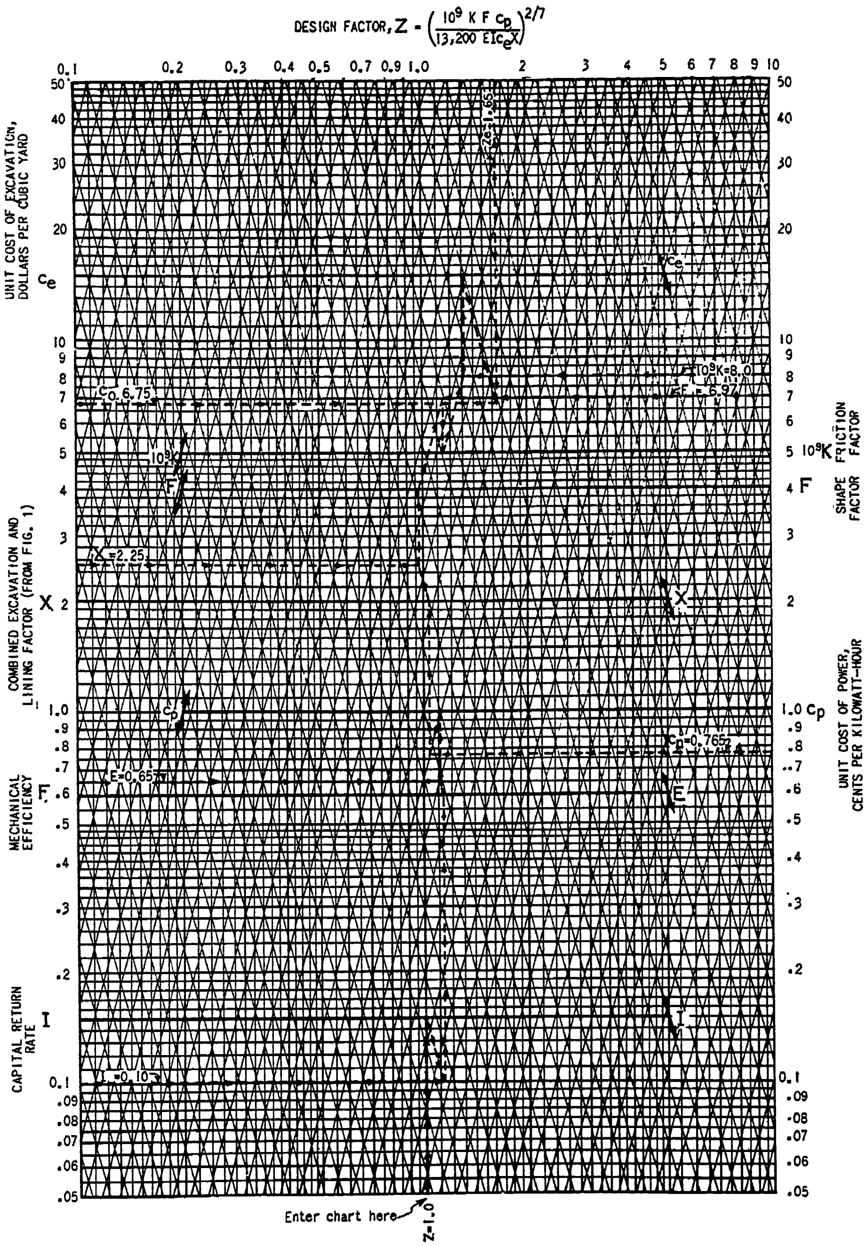

Fig. 2.—Proportionality chart for determining design factor Z for use in formulas and charts for determining economical size of openings used as mine airways.

Unit cost of power, in cents per kilowatt-hour, is power-cost rate times ratio of operating time for discontinuous operation.

line at the bottom, which is followed to intersection with the first base line I, where the change to actual value of I is made parallel to the proper diagonal ruling, as indicated by the flag on the right end of the I base line; the new value of Z is then transferred vertically to the next base line above and the procedure is repeated of alternately passing parallel to the proper diagonal ruling, as indicated by the flag on the base line, to actual factor value and then transferring the new value of Z vertically to the next base line in order, and finally to the scale on the upper edge where the value of Z that satisfies all of the factor values is read off. The example indicated by dotted lines shows that when $I = 0.10$, $E = 0.65$, $c_p = \$0.765$, $X = 2.25$, $F = 6.97$, $K = 0.0^{8}8$, and $c_e = \$6.75$, $Z = 1.65$.

By similar separate steps, the effect of a change in the value of any of the separate factors on the value of Z, and thus on the area for constant quantity, may be determined as a ratio. One of the values considered is used as a base value for that factor, and from its intersection with $Z = 1$ a line is passed, parallel to the proper diagonal ruling, to intersection with the second value assumed for the same factor. This determines a new value for Z, and its comparison with $Z = 1$ determines directly the ratio of the change that would be affected in any other value of Z by the same relative change in the factor considered.

Chart for Economic Area, A_m

With Z known, the next step is to determine the economic area for a particular quantity of flow. This can be accomplished graphically by means of the chart presented as Fig. 3. The method of using the chart is shown by the dotted line example. The chart is entered with the previously determined value of Z at the bottom; its intersection with the proper quantity line on the diagonal scale determines the area, read on the scale on the left. The example shows that when $Z = 1.65$ and $Q = 100{,}000$ cu. ft. per minute, $A_m = 85.4$ square feet.

The effect of changes in quantity of flow on economic area may also be examined by means of this chart, by assuming a constant value for Z and noting the ratio of change in area that accompanies any assumed change in quantity.

Chart for Determining Costs When Area Is That Required for Maximum Economy

With the economic area known, the next step is to determine the total yearly cost C and its component parts, yearly power cost C_p, and yearly capital charge, C_c. It may also be desirable to determine the component parts of the yearly capital charge, as that due to excavation, C_e, and that

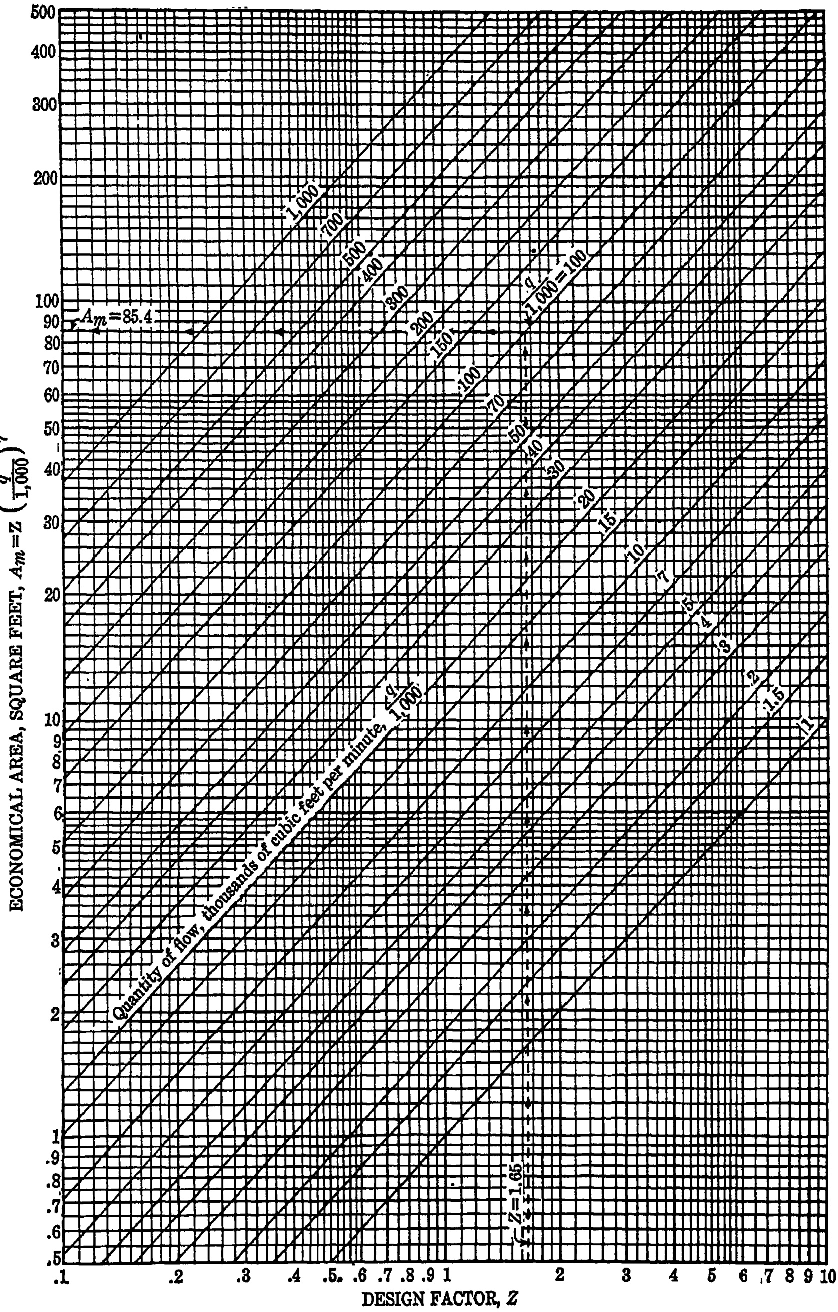

FIG. 3.—CHART FOR DETERMINING ECONOMICAL AREA A_m OF OPENINGS USED AS MINE AIRWAYS. DESIGN FACTOR Z FROM FIG. 2.

due to lining, C_s. In all cases, the costs will be a direct function of the length, and it is convenient, therefore, to consider only costs per linear foot. In preparing a chart for the graphic determination of these costs, presented as Fig. 4, the following relations were used:

$$C_e = IA_mc_eG;\ C_s = IA_mc_sJ;\ C_c = C_e + C_s = IA_mc_eX$$
$$\text{and } C_p = 0.4C_c;\ C = C_c + C_p = 1.4C_c.$$

The unit costs C_e and C_s are in dollars per cubic foot, but it is considered more convenient to use unit costs in dollars per cubic yard in the chart, so that the right-hand member of each of the first group of three equations must be divided by 27 to correspond to these units.

The method of using the chart is shown by the dotted line example. The value of X, entered on the lower scale, is carried vertically to intersection with the diagonal scale value of A_m, then horizontally to the "1 cu. yd. = 27 cu. ft." line, then vertically to the diagonal scale value of c_e, then horizontally to intersection with the diagonal scale value of I, and then vertically both to the upper scale, where the C_c value is read, and to intersection with the "$C = 1.4C_c$" and "$C_p = 0.4C_c$" lines. The scale values of C and C_p can be read by projecting these two points of intersection horizontally to the scale on the left.

C_e is determined in the same manner as C_c, except that the chart is entered on the G scale value on the bottom instead of X. C_s is also determined in the same manner as C_c, except that the chart is entered with the value of J (equal to $G - 1$) and c_s is used instead of c_e.

The example shows that when $X = 2.25$, $A_m = 85.4$, $c_e = \$6.75$, and $I = 0.10$, $C_c = \$4.82$, $C_p = \$1.93$ and $C = \$6.75$. To avoid confusion, solutions for C_e and C_s have not been indicated on the chart, but if, for the same example, $G = 1.625$ and $c_s = \$6.75$, it will be found that $C_e = \$3.48$ and $C_s = \$1.34$.

Economical Size of Fan Tubing

In the use of tubing and pipe for auxiliary ventilation in mines and tunnels, velocity pressure losses, due to high-velocity discharge, deflections from a straight line and occasional area changes, are often as important as friction pressure losses in determining power requirements. In industrial installations it is claimed that the thickness of pipe used generally varies about as the square root of the diameter of the pipe, but an examination of reported costs for mine installations indicates that the thickness of the material used varies more nearly as the diameter—at a slightly higher rate for canvas and jute tubing and at a slightly lower rate for galvanized-iron pipe. The same assumption that has been made for mine airways—that the thickness of the lining material varies directly as the diameter—is therefore sufficiently exact for fan tub-

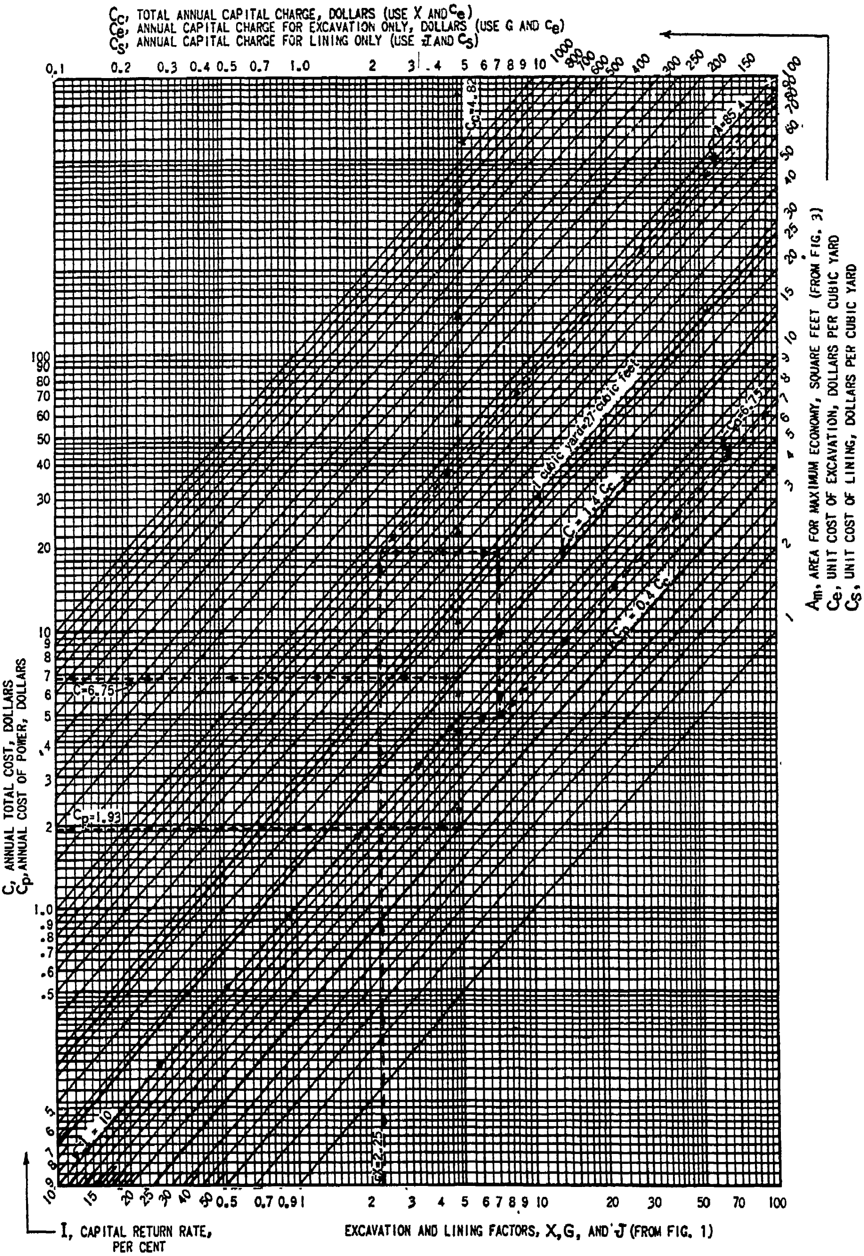

FIG. 4.—CHART FOR DETERMINING ANNUAL CAPITAL CHARGES, POWER COST AND TOTAL COST PER LINEAR FOOT FOR FLOW OF AIR THROUGH A MINE OPENING OF THE SIZE REQUIRED FOR MAXIMUM ECONOMY OR MINIMUM TOTAL COST.

ing and fan pipe. The formula based on friction pressure losses (see Appendix 1) only is

$$\frac{(A_c)^{7/2}}{(A_m)} = 0.4\frac{C_{sc}}{C_{pe}} \text{ or } A_m = \frac{A_o}{0.770\frac{(C_{so})^{2/7}}{(C_{po})}}$$

and the relation is such that, when the area for maximum economy is used, the yearly costs are 40 per cent of the yearly capital charges.

The corresponding formula based on velocity pressure losses only (see Appendix 2) is

$$\left(\frac{A_o}{A_m}\right)^3 = 0.5\frac{C_{so}}{C_{po}}$$

and the relation is such that, when the area for maximum economy is used, the yearly power costs are 50 per cent of the yearly capital charges.

While a correct solution for a particular case would require a compromise between the two areas, as found through the two different formulas, based on the estimated proportion of each set of pressure losses to the total pressure losses, the formulas are quite similar and the difference in results for economic area and costs for any particular case are small. In addition, there are many indeterminates in fan-tubing installations that cannot be treated mathematically, fixed nominal sizes only are readily procurable, and the total costs involved are relatively small, so that the use of either formula alone to give an approximate solution is justified. The particular conditions of the installation might be allowed to dictate which formula should be used. The friction formula permits graphic solutions by means of the charts previously presented, whereas the velocity formula is more easily handled in slide-rule computations. Fan-installation costs should be considered as entirely independent of the tubing costs, even though they are casually related.

APPENDIX 1.—Derivation of Simple Formula for Economic Area Based on Friction Pressure Losses

The pressure losses in an airway are the sum of the friction pressure losses and the velocity pressure losses, or

$$H = H_f + H_v \tag{1}$$

where H = total pressure loss in inches of water,
H_f = friction pressure losses,
H_v = velocity pressure losses.

The general equation for friction pressure losses on a quantity basis is

$$H_f = \frac{KpLQ^2}{5.2A^3} \tag{2}$$

where K = friction factor for foot-pound-minute units,
p = perimeter, feet,
L = length, feet,
Q = quantity of flow, cubic feet per minute,
A = cross-sectional area of airway, square feet for free area for flow.

For constant shape of airway, perimeter varies directly as the square root of the area, or

$$p = FA^{1/2}, \tag{3}$$

where F = shape factor, a constant for any particular shape of airway, so equation 2 may be written

$$H_f = \frac{KFLQ^2}{5.2A^{5/2}}. \tag{4}$$

With K, F, L and Q constant, and with the subscript o used to denote values corresponding to a particular area, A_0, we have, from equation 4

$$\frac{H_f}{H_{fo}} = \left(\frac{A_o}{A}\right)^{5/2}. \tag{5}$$

With Q constant, power requirements, and therefore yearly power cost, C_p, vary directly as the pressure losses, or

$$\frac{C_p}{C_{po}} = \frac{H_f}{H_{fo}} = \left(\frac{A_o}{A}\right)^{5/2}. \tag{6}$$

The yearly capital charge for excavation, C_e, and for lining, C_s, will be a constant proportion of the total cost, which will vary directly with the total volume of excavation and lining. With length constant, total volumes vary directly with the cross-sectional areas, and the latter, when the lining thickness is a constant ratio of the dimensions, vary directly as the free area, so that

$$\frac{C_e}{C_{eo}} = \frac{A}{A_o}, \text{ and } \frac{C_s}{C_{so}} = \frac{A}{A_o}. \tag{7}$$

The yearly total cost C is the sum of the yearly power cost and the yearly capital charges for excavation and lining, or

$$C = C_p + C_e + C_s = C_{po}\frac{A_o^{5/2}}{A^{5/2}} + C_{eo}\frac{A}{A_o} + C_{so}\frac{A}{A_o}. \tag{8}$$

The yearly total cost will be a minimum when the first differential is equal to 0; that is, when

$$-\frac{5}{2}\frac{C_{po}A_o^{5/2}}{A^{7/2}} + \frac{C_{eo}}{A_o} + \frac{C_{so}}{A_o} = 0. \tag{9}$$

Substituting A_m for A in the foregoing as representing the area for maximum economy, and simplifying, we have

$$\left(\frac{A_o}{A_m}\right)^{7/2} = 0.4\,\frac{C_{eo} + C_{so}}{C_{po}} \qquad [10]$$

and

$$A_m = \frac{A_o}{0.770\left(\dfrac{C_{eo} + C_{so}}{C_{po}}\right)^{2/7}}. \qquad [11]$$

APPENDIX 2.—Derivation of Simple Formula for Economic Area Based on Velocity Pressure Losses

The general equation for velocity pressure losses is

$$H_v = Xh_v \qquad [12]$$

where X = number of velocity pressures lost in airway,
h_v = pressure corresponding to average velocity.
Then, since velocity pressure varies as the square of the velocity V and area varies inversely as the velocity,

$$\frac{H_v}{H_{vo}} = \left(\frac{V}{V_o}\right)^2 = \left(\frac{A_o}{A}\right)^2. \qquad [13]$$

Yearly power costs vary directly with pressures for constant quantity, and

$$\frac{C_p}{C_{po}} = \frac{H_v}{H_{vo}} = \left(\frac{A_o}{A}\right)^2;\ C_p = C_{po}\left(\frac{A_o}{A}\right)^2.$$

From equation 7:

$$C_e = C_{eo}\frac{A}{A_o} \text{ and } C_s = C_{so}\frac{A}{A_o}; \qquad [14]$$

then

$$C = C_p + C_e + C_s = C_{po}\left(\frac{A_o}{A}\right)^2 + C_{eo}\frac{A}{A_o} + C_{so}\frac{A}{A_o}. \qquad [15]$$

As before, the total cost will be a minimum when the first differential is equal to 0; that is, when

$$-\frac{2C_{po}A_o^2}{A^3} + \frac{C_{eo}}{A_o} + \frac{C_{so}}{A_o} = 0. \qquad [16]$$

Substituting A_m for A as representing the area for maximum economy, and simplifying,

$$\left(\frac{A_o}{A_m}\right)^3 = 0.5\,\frac{C_{eo} + C_{so}}{C_{po}} \qquad [17]$$

and

$$A_m = \frac{A_o}{0.794\left(\frac{C_{eo} + C_{so}}{C_{po}}\right)^{1/3}}. \qquad [18]$$

APPENDIX 3.—Derivation of General Formula for Economic Area Based on Friction Pressure Losses

With the same nomenclature as heretofore used, and with such additional designations as are given,

$$C_p = \frac{c_p \times \text{air horsepower}}{E} \qquad [19]$$

where c_p = unit cost of power per horsepower-year, in dollars,
E = over-all mechanical efficiency of power-using unit as a ratio.

$$\text{Air horsepower} = \frac{PQ}{33{,}000} \qquad [20]$$

where P = pressure in pounds per square foot = 5.2 times pressure in inches of water.

From equation 4:

$$P = 5.2H_f = \frac{KFLQ^2}{A^{5/2}}. \qquad [21]$$

Therefore

$$C_p = \frac{c_pKFLQ^3}{33{,}000EA^{5/2}}. \qquad [22]$$

$$C_e = Ic_eAGL \text{ and } C_s = Ic_sAJL \qquad [23]$$

where I = yearly capital return rate as ratio of total first cost, for interest on, and amortization of, the capital invested in the airway,
c_e = unit cost of excavation in dollars per cubic foot,
c_s = unit cost of lining in dollars per cubic foot of space occupied,
G = ratio of total excavation area to free area, termed "excavation factor,"
J = ratio of area occupied by lining to free area, termed "lining factor."

Then

$$C = C_p + C_e + C_s = \frac{c_pKFLQ^3}{33{,}000EA^{5/2}} + Ic_eAGL + Ic_sAJL. \qquad [24]$$

As before, the total yearly cost C will be a minimum when the first differential is equal to 0; that is, when

$$-\frac{5}{2}\frac{c_pKFLQ^3}{33{,}000EA^{7/2}} + Ic_eGL + Ic_sJL = 0. \qquad [25]$$

Substitute A_m for A in the foregoing as representing the area for maximum economy, multiplying the first term by $\frac{10^9}{10^9}$ for convenience in computation, and simplifying,

$$A_m = \left(\frac{10^9 KFc_p}{13{,}200EI(c_eG + c_sJ)}\right)^{3/4}\left(\frac{Q}{1000}\right)^{9/4}. \qquad [26]$$

APPENDIX 4.—Derivation of Formulas for Excavation Factor G Without Allowance for Overbreak

The excavation factor in the previous formula is $\frac{\text{area outside lining}}{\text{free area}}$.

In terms of free area dimensions, let

O = length of diameter of circle, distance across flats of regular polygon, side of square, or shorter side of rectangle,

M = ratio of longer to shorter side of rectangle,

R = ratio of thickness of lining to span, or to longer side of rectangle,

N = number of similar compartments of equal size in compartmented airways.

In compartmented airways, the dividers are assumed to have the same thickness as the wall lining.

For a circle,

$$G = \frac{\frac{\pi}{4}(O + 2RO)^2}{\frac{\pi}{4}O^2} = (1 + 2R)^2. \qquad [27]$$

For a regular polygon, where u is the angle subtended at the center by one side and n is the number of sides,

$$G = \frac{\frac{(O + 2RO)^2}{4}\left(n \tan \frac{u}{2}\right)}{\frac{O^2}{4}\left(n \tan \frac{u}{2}\right)} = (1 + 2R)^2. \qquad [28]$$

For a square,

$$G = \frac{(O + 2RO)^2}{O^2} = (1 + 2R)^2. \qquad [29]$$

For a rectangle,

$$G = \frac{(O + 2ROM)(OM + 2ROM)}{O^2M} = (1 + 2R)(1 + 2RM). \qquad [30]$$

For N rectangular compartments in line,

$$G = \frac{[OM + 2ROM][NO + (N + 1)ROM]}{NO^2M}$$
$$= \frac{N + NRM + RM + 2NR + 2NR^2M + 2R^2M}{N}$$
$$= (1 + 2R)\frac{(N + RM + NRM)}{N}. \qquad [31]$$

For a rectangle lined on three sides only, with shorter side not lined,

$$G = \frac{(O + 2ROM)(OM + ROM)}{O^2M} = (1 + R)(1 + 2RM); \qquad [32]$$

with longer side not lined,

$$G = \frac{(O + ROM)(OM + 2ROM)}{O^2M} = (1 + 2R)(1 + RM). \qquad [33]$$

Propeller Type Mine Fan at Moose Shaft, Butte, Montana

By A. S. Richardson,* Butte, Mont.

(New York Meeting, February, 1932)

The recent installation of a high-pressure propeller type fan at the Moose shaft of the Anaconda Copper Mining Co. at Butte, Mont., is of interest on account of novelty of design and also because an appreciable saving was made over the cost of a reversible centrifugal fan such as is ordinarily used where a high pressure is required. The lower cost is due solely to the fact that by means of controls on the power supplied to the three-phase, alternating-current, induction motor, directly connected to the propeller it is possible to reverse the direction of rotation of the propeller and thus reverse the direction of air flow. The extra air ducts and reversing doors required for this purpose with centrifugal fans are not necessary, and their elimination greatly reduces these construction charges, which form a great part of the total expense.

The Badger State mine for which the Moose shaft is used as the main outlet air course is now being developed to the 4100-ft. level. The old centrifugal fan, formerly used at the collar of the shaft, was installed 15 years ago at a time when most of the stoping was being done above the 2400-ft. level, and when the temperature of the rock was much lower than it is now on the bottom levels recently opened. The old fan, therefore, became inadequate to maintain desirable ventilation conditions, on account of the increase in resistance to air flow incident to the greater depth of the mine workings and because a much larger volume of air is required to offset the higher rock temperatures.

Propeller, or disk, fans commonly used in the ventilation of mines are not suited to high-pressure work, but economies in the cost of their installation have long been recognized. In South Africa use has been made of a number of propellers placed in series along a common shaft, the number of propellers used being varied with requirements for meeting different conditions of mine resistance, but the mechanical efficiency is described as being low.

In Butte, experimental work was done by the writer along similar lines under somewhat different conditions, but with poor results as to both pressure capacity and mechanical efficiency. Tests were also made of two different types of propeller fans of moderate size, which were said by the manufacturers to be suited to high-pressure mine ventilation.

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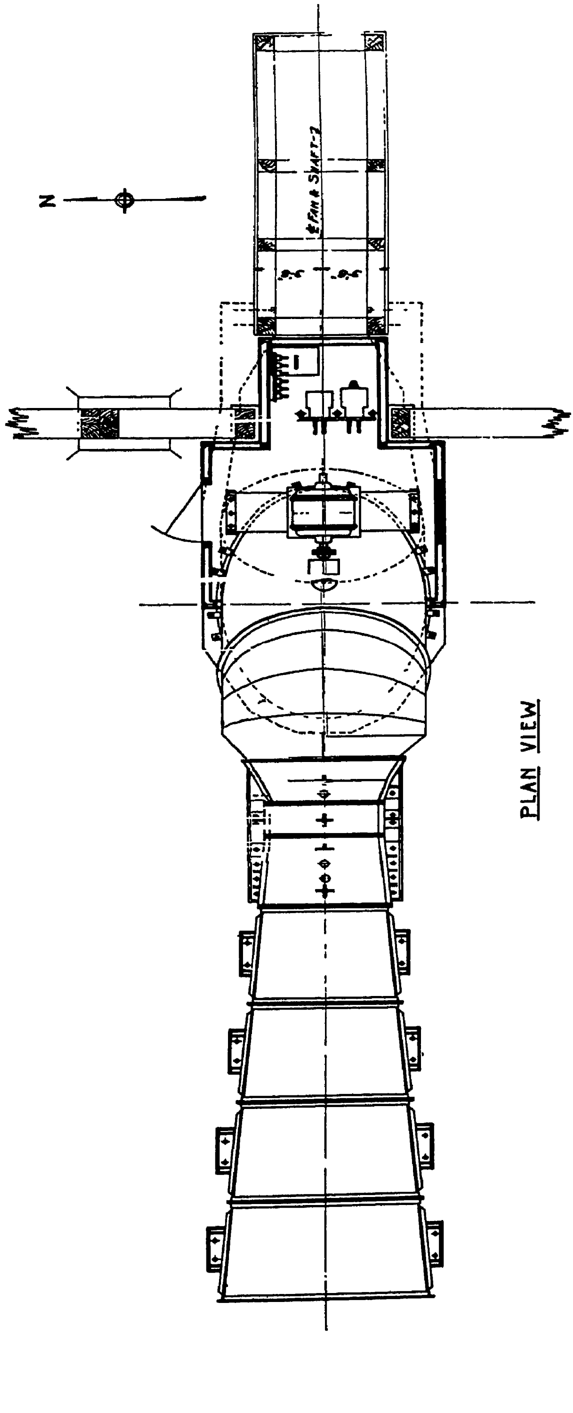
One of these propellers had three blades so mounted in the hub that each blade could be rotated around its own axis, thus making it possible to set the blades at any desired pitch, as necessary under different conditions as to mine resistance. Actual performance of this propeller was less than one-third the rated capacity given by the manufacturer and it was capable of developing only a very limited pressure. The other propeller tried had much better pressure capacity, but was also much overrated by the manufacturer, and required considerable repair work to keep it running; so that it was not a desirable piece of equipment.

C. D. Woodward, chief engineer for the Anaconda Copper Mining Co., was favorably impressed with reports about a propeller type fan designed by H. F. Schmidt, of the Westinghouse Electric and Manufacturing Co., and through his interest an investigation was undertaken to determine whether or not it might be used at the Moose shaft. Tests conducted at the Westinghouse plant, witnessed by the writer, showed that with this type fan, the weak points of the common disk fan had been eliminated and the operating characteristics were well suited to mine-ventilation service.

An account of the theory and experimental work upon which the design of the fan was based was published in the *Journal* of the American Society of Naval Engineers, February, 1928. Briefly stated, it is that the rotation of the blades of a screw propeller is the equivalent of moving a disk of the same area in free water, and that a jet is caused to flow by this action similar to the jet produced by flow through a sharp-edged orifice. From this it follows that since the area of the vena contracta is 0.625 that of the sharp-edged orifice, maximum efficiency will obtain when the projection of the propeller blades upon a plane normal to the axis of rotation is 0.625 that of the area of the circle described by the rotation of the propeller. Also, that since the velocity of flow through the vena contracta is 1.6 times the velocity of flow through the sharp-edged orifice, it is necessary that the pitch of the propeller blades should increase from the leading to the trailing edge in the ratio of 1.6. A large part of the experimental work was done by the use of air propellers.

The fan installed at the Moose shaft was designed to exhaust 300,000 cu. ft. of air per minute against a mine resistance equivalent to 9 in. of water, with air having a density of 0.057 lb. per cubic foot. The propeller is 72 in. dia. The pitch ratio at the inlet edge is 1, while the pitch ratio at the discharge edge is 1.6. Guide vanes are provided on both inlet and discharge sides of the propeller, to prevent rotational currents in the air. Power required, under the conditions stated, is 565 hp. and the propeller is direct connected to a 700-hp., 1770 r.p.m. motor.

Plans for the installation were made under the supervision of F. C. Jaccard, mechanical superintendent. The general arrangement and main dimensions are shown on Figs. 1 and 2. Connections between fan inlet and mine shaft are made so that all shaft compartments are acces-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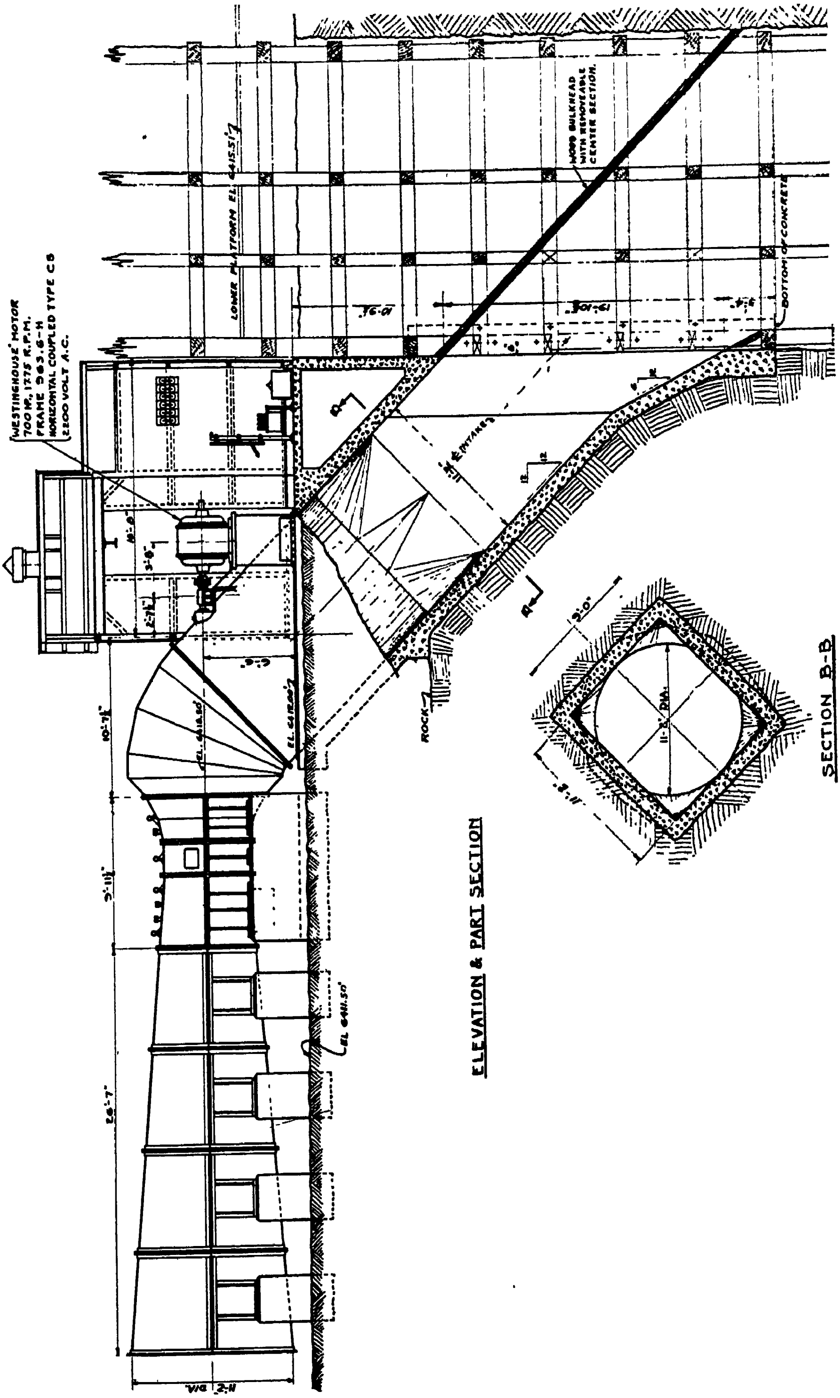


FIG. 1.—GENERAL ARRANGEMENT OF FAN.

sible from the collar and may be used as necessary for repairs or emergency purposes.

The manner in which turns in the air course, and changes in cross-sectional area, are made at such a place has a great bearing on the resistance to air flow, and effort was made to reduce this to a minimum. It was necessary to run the air duct from one end of the shaft, or an oblong opening, making a turn of 90°, and to connect to the circular fan intake. To accomplish this, it was necessary to make a number of changes in cross-sectional area, which had to be done without abrupt transition and called for considerable skill in concrete work, both as to forms and handling of materials. Guide vanes are placed in the elbow to reduce resistance to flow at this point.

The discharge stack, which is a very inefficient attachment to most centrifugal fans, functions very well with this propeller. Velocity of flow is reasonably uniform at all points across the discharge end and there are practically no rotational currents.

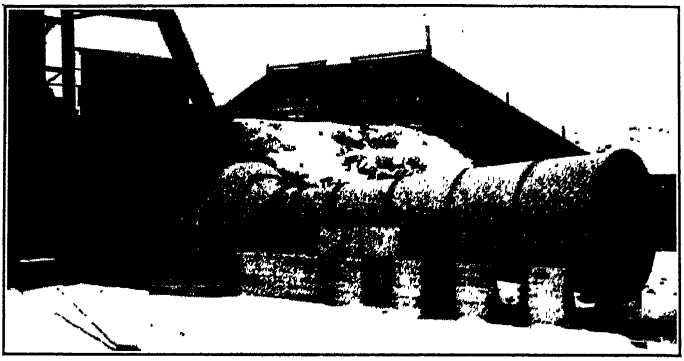


FIG. 2.—OUTLET OF FAN.

On account of the high speed at which the propeller is driven, special care was taken to guard against trouble with the bearings. Lubrication of the main propeller bearings is effected by means of a pump, and a pressure relay is carried from the oil circuit to the no-voltage release. In addition to this, all five bearings, on motor, stub shaft and propeller are equipped with thermal relays, which stop the fan in case of overheating.

Reversal of the direction of rotation, or air flow, is accomplished by the use of a double-throw oil circuit breaker and the time required for the operation is one minute. With centrifugal fans it is generally necessary to enter the air ducts to throw some of the doors and, unless care has been taken to keep them in good working order, it is probable that corrosion may have rendered them difficult to move—an undesirable condition when quick action is necessary and the air ducts are full of smoke or gas.

A favorable operating feature of this propeller fan is that when the resistance of the mine workings increases the fan automatically develops more pressure when running at constant speed, and thus tends to maintain the circulation at the expense of an increase in power consumption. The common type of centrifugal fan with forward-tipped blades develops but a slight increase in pressure under the same circumstances, and the circulation of air and power consumption are both subject to a marked decrease. Mechanical efficiency varies somewhat similarly under these

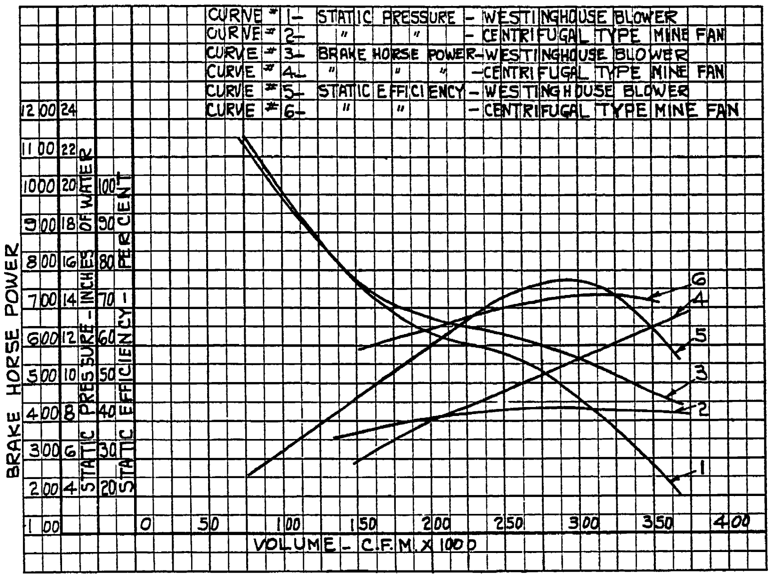


FIG. 3.—OPERATING CHARACTERISTICS OF WESTINGHOUSE BLOWER AND CENTRIFUGAL TYPE MINE FAN.

conditions with both types of fans, and it is, of course, generally possible to change pulleys where a belt drive is used and in this way to speed up a centrifugal fan. This, however, is a matter of some expense, and when the fan is direct connected to an alternating-current induction motor a satisfactory solution to the problem is more difficult. Fig. 3 shows the operating characteristics of both types of fans.

As previously stated, savings in cost of installation are due entirely to the elimination of ducts and doors required for reversing the air flow with centrifugal fans, and the amount of such savings will vary considerably under different conditions and with different types of construction. In the present case the cost of the propeller fan was 60 per cent greater than the cost of a centrifugal fan of equivalent capacity, but the total cost of the completed installation was 40 per cent less than it would have been had the centrifugal fan been used.

Pressure required of this, and a number of other main surface fans in the Butte district, is somewhat higher than average because the depth of the operating zone in certain mines is now lower than the 3000-ft. level. Ventilation depends very largely upon the air-carrying capacity of the outlet air shafts which were originally intended for use as operating shafts and are of rather small size because of the ground conditions. Natural rock temperatures range up to 120° F., so that large volumes of air are required and power consumption is relatively of minor importance. To reduce frictional resistance of the air shafts to a minimum, the old square-set shaft compartments have been smooth-surfaced and extensions of the shafts to greater depths have been made by octagonal raises on skin to skin timber.

From the surface to the 1600-ft. level the Moose shaft is vertical and has three compartments, one compartment 7 ft. 6 in. by 5 ft. 0 in. and two compartments 4 ft. 8 in. by 5 ft. 0 in. All three compartments are smooth-surfaced to reduce frictional resistance. Below the 1600-ft. level an octagonal opening has been substituted for the rectangular three-compartment shaft. It is supported on skin to skin timber and measures 9 ft. 0 in. on the perpendicular between timber faces. From the 1600 to 2000-ft. levels the octagon is vertical, but from the 2000 to the 3000-ft. level it makes an angle of 53° with the horizontal. Below the 3000-ft. level, the dip will be 68°, or roughly parallel to that of one of the main productive veins; an arrangement that reduces the length of connecting crosscuts to a minimum.

In a number of instances, the main surface fans that were installed some years ago are not now well adapted to maintain the required air circulation, because of the increase in resistance to flow due to the greater depth at which operations are now conducted. To provide additional pressure, or power, it has, therefore, been necessary to make increasing use of booster fans in the underground air circuits. Although the use of such fans is often a matter of practical necessity, it is generally more desirable to control the ventilation system directly from the main surface fan, provided this can be done. At the Badger mine, where the Moose shaft is situated, the installation of the new fan afforded an opportunity to provide for higher pressure, and two booster fans formerly in use have been taken out of service. Future increase in mine resistance will be offset to a large extent by the increase in pressure automatically effected by the fan.

On account of the fact that important changes were made in the main air courses, including completion of smooth-surfacing of the air shaft, it was necessary to base the estimate as to pressure requirements upon calculations from factors of frictional resistance. In actual running tests it has been shown that the fan draws approximately 290,000 cu. ft. of air per minute from the mine against a resistance equivalent to 9.8 in. of

water and with a power consumption of 570 hp. The writer has never considered mine air measurements to be anything other than working approximations, but the agreement between estimated and actual working results is somewhat closer than is usually realized. Extension of the Moose shaft to greater depth will provide additional air inlets to the shaft and reduce the total mine resistance with a slight increase in volume of air circulated.

Full benefit in improvement of ventilation conditions resulting from the operation of the fan has not yet been realized but the average temperature of the working places such as stopes, crosscuts, drifts and raises has already been lowered approximately 5° F. At time of writing, only five places in the mine showed a psychrometer wet bulb higher than 80° F. and velocity of air movement has been greatly increased, which, of course, increases the cooling power of the air. Future changes in the ventilation system will improve distribution of the air with further benefit to the active working zone.

DISCUSSION

H. F. Schmidt, Philadelphia, Pa. (written discussion).—Mr. Richardson is to be complimented for his excellent paper directing attention to a new development in mine ventilation.

To Mr. Woodward, chief engineer, and to Mr. Richardson should be given the credit for having undertaken a new development in adopting the propeller type fan for so important an installation, since this type of fan for many years has been known only as a very inefficient means for moving large volumes of air against low resistance, whereas the present installation was designed for 9-in. static pressure at a barometer of 23½ inches.

As Mr. Richardson says, the characteristics of the propeller type fan are ideal for mine ventilation because of the steep pressure volume characteristic, which permits the fan to deliver nearly the same volume even though the mine resistance be materially increased by later extensions over that originally contemplated.

Because this type of blower operates at much higher speed than is generally employed for mine ventilation, some question may arise concerning the stresses involved and also the effect of erosion at the relatively high tip speeds—approximately 550 ft. per second in the case of the Moose fan.

The propellers employed in these fans have blades tapered from the root to the tip. They are made of steel forgings and are completely machined to assure accuracy of contour and uniformity of thickness and taper over the entire blade. These blades are warped to shape and then welded into grooves in the hub. The taper is so designed that the stress at 10-in. static pressure is 6000 lb. per sq. in. at the root of the blade and the stress in the welding is approximately 4000 lb. per sq. in. As the stress varies directly as the static pressure, the stress would be only 35,000 lb. per sq. in. for 40-in. static pressure.

In regard to erosion, the same question arose in connection with the first centrifugal blowers installed for blast-furnace work. However, the first blower installed by the Westinghouse Machine Co. at the Great Falls plant 20 years ago was recently moved to the plant at Anaconda. This blower had a tip speed of 600 ft. per second and has

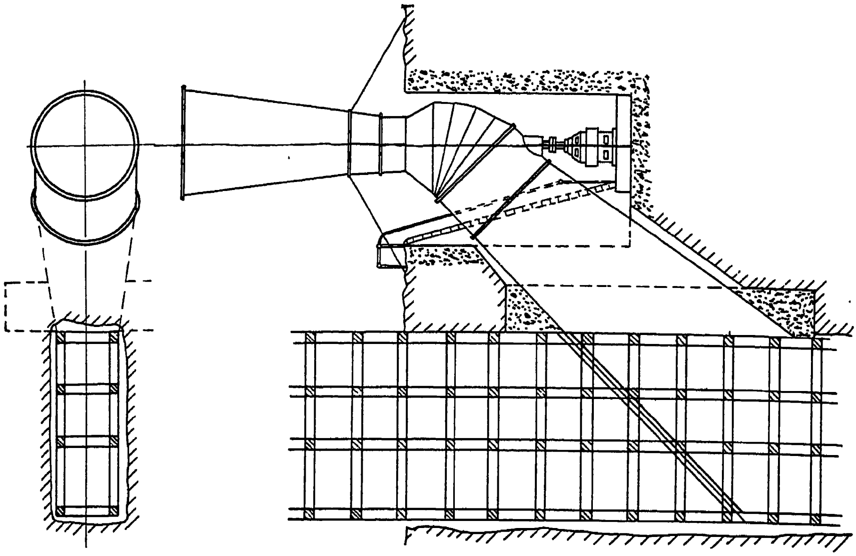


Fig. 4.—Vertical arrangement of fan.

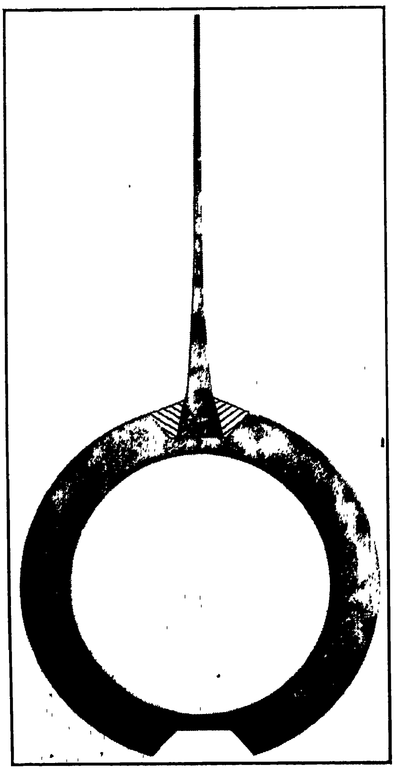


Fig. 5.—Cross-section through blade.

Fig. 6.—Arrangement and slope of guide vanes on outlet side of propeller.

shown no noticeable wear in 20 years of service; consequently, no more anxiety need be felt in regard to the erosion of the blades of propeller blowers. The blades of the Moose fan are 2½ in. thick at the root and ⅜ in. thick at the tip, a far stronger construction than any heretofore employed in ventilating blowers.

Mr. Richardson has mentioned that in deep mines with high resistance the cost of power is of minor importance. This will be understood by mining engineers in the sense in which Mr. Richardson intended it; namely, that unless the workings are kept at a reasonable temperature the output per man is so reduced that the mine may become unprofitable. However, the higher the resistance in the mine, the greater the power input per thousand cubic feet of air, and where power rates are high the efficiency of the ventilating blowers becomes of considerable importance and a difference of a few per cent in efficiency will offset a very large difference in the initial cost of the installation.

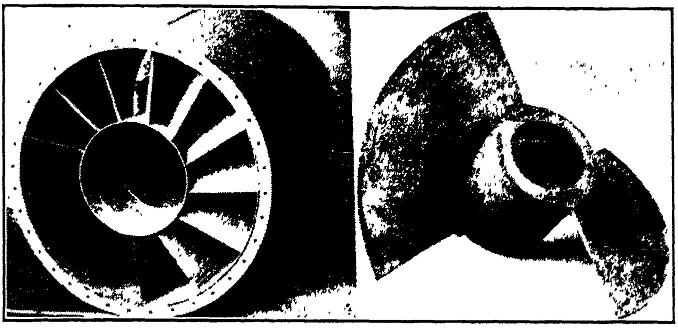
7  8

FIG. 7.—GUIDE VANES FOR BLOWER FURNISHING 75,000 CU. FT. OF AIR PER MINUTE.
FIG. 8.—PROPELLER FOR HELL GATE BLOWER BEFORE WELDING WAS GROUND SMOOTH.

The writer feels that Mr. Richardson did a remarkably good piece of work in estimating the resistance of the Moose shaft, as indicated by the very close agreement between the estimated resistance and that actually measured after the installation of the fan.

Although the horizontal blower arrangement was selected for the Moose shaft, the writer feels that in general a vertical arrangement, shown in Fig. 4, is slightly more efficient and considerably cheaper to install because of the smaller foundations required. Also, the vertical arrangement is somewhat more accessible.

In reply to a question from the floor, as to whether this fan was the outgrowth of the disk fan or the airplane propeller, Mr. Schmidt said that it was a new departure in fan design along lines he had thought out years ago. It is a two-blade propeller, both blades together having a projected area equal to 62.5 per cent of the disk area. The static efficiency of the Butte installation is 79 per cent at a static pressure of 9.8 in. of water. With a velocity head of 0.45 in. this gives a total pressure of 10.25 in. and a total efficiency of 83.5 per cent, which is considerably higher than for any other mine fan of which he knows.

As to the static pressure capacity of fans of this type, three such blowers have been operating continuously for over three years at the Hell Gate power plant, each delivering 100,000 cu. ft. of air per minute at 16.5-in. water gage. Their static efficiency is 78 per cent and their total efficiency 83 per cent. They could be operated satisfactorily up to 30-in. water gage, or beyond. At 40-in. water gage, the stress at the root of the blade reaches the safe limit.

Asked about the shape of the blade, Mr. Schmidt said that it was designed for uniform stress. A cross-section through the blade is shown in Fig. 5, which also shows the welding. This parabolic section is that existing at all elements normal to the axis of rotation and is proportioned to give the minimum stresses at all points as well as to give the proper resistance to vibration.

The arrangement and slope of the guide vanes on the outlet side of the propeller is shown in Fig. 6 in their correct relative positions. While these are actually for a pump, they are absolutely the same as those used for blowers. Guide vanes for a blower furnishing 75,000 cu. ft. of air per minute are illustrated in Fig. 7.

A question was asked about centrifugal effects and Mr. Schmidt replied that the air shows no tendency to rotate before entering the blades of the propeller, so that inlet guide vanes are unnecessary. The guide vanes on the outlet side of the propeller used are for the bearing supports. The thrust is taken up by ball bearings.

In response to a question about dividing the propeller into more blades, Mr. Schmidt said that the edge losses increase 3 to 3.5 per cent for each blade. In a propeller type blower the function of the propeller is only to impart velocity to the air, and the pressure created results from converting the velocity head imparted into static pressure. This is done by a diverging tube or diffuser and by properly curved guide vanes. The efficiency of the propeller as a means of imparting velocity to the air may be inferred from the fact that the maximum efficiency of a Venturi tube is not over 95 per cent and with the curved guide vanes probably not over 90 per cent. Therefore, if the total efficiency is found to be 83 per cent this implies an efficiency of about 92.3 per cent for the two-blade impeller. If this were made a four-blade propeller its efficiency would be further reduced about 7 per cent because of additional edge losses.

Asked about the effect of skin friction, Mr. Schmidt said that not all of the 7 per cent loss cited would be edge loss, strictly speaking, as some of it would be due to surface friction, although, on a ship, it is known that the skin friction per foot is two or three times as great in the first foot as it is farther back. The mean velocity relative to a surface is less for a long surface than for a short one.

Fig. 8 shows one of the propellers for the Hell Gate blowers before the welding was ground smooth.

A. S. Richardson (written discussion).—Mr. Schmidt says that "the air shows no tendency to rotate before entering the blades of the propeller so that inlet guide vanes are unnecessary." This statement is certainly correct when the propeller is drawing air from a large chamber under usual test conditions. As installed at the Moose shaft such conditions do not exist, and there is a very strong eddy current at the propeller inlet, which is, of course, largely due to the elbow immediately preceding it. Evidence of this is clearly shown by markings on the inside of the inlet cone that are caused by air. Similarly, in connection with reversible centrifugal fans, as commonly installed, the horizontal projection of the air course through the fan shows that the air makes a U-turn, and it is evident that a large part of the total air volume, probably 60 per cent, short-circuits directly through the fan wheel into the discharge stack without entering the housing. These also are conditions that do not exist under test regulations, and which have a bearing on fan performance that is not taken into account.

The statement that in a propeller type blower the function of the propeller is only to impart velocity to the air and the pressure created results from converting the velocity head imparted into static by means of diffuser and guide vanes is of interest, and a few calculations will show that it is correct so far as the Schmidt propeller fan is concerned. However, I have at hand a design for a propeller fan capable of developing more than 6 in. static pressure at maximum efficiency, and there are no guide vanes or diffuser. The design was, I believe, prepared by Mr. Schmidt, and there are other propellers which develop pressures equally high. Some remarks from Mr. Schmidt in explanation would be of interest.

The System PbO-Sb_2O_3 and Its Relation to Lead Softening*

By C. G. Maier† and W. B. Hincke,‡ Berkeley, Calif.

(New York Meeting, February, 1932)

Commercial processes of lead softening directly involve the behavior on fusion of mixtures of the oxides of antimony and lead, and the vapor pressures of these materials. Practically no quantitative data have been available for the discussion of lead softening from the point of view of the fundamental metallurgy of the process. The Pacific Experiment Station of the United States Bureau of Mines, in cooperation with the University of California, in the course of its program of study of the fundamental properties of oxides and sulfides, recently made vapor pressure measurements upon the various forms of pure antimony trioxide.[1]

When these measurements were considered in connection with the softening and "fuming" furnaces which are often used for lead purification, an unexpectedly anomalous condition appeared. In brief, this anomaly consisted in the fact that in certain softening furnaces, where lead bullion of approximately 1 per cent antimony content was being treated at a temperature of approximately 1300° F., no appreciable volatilization of antimony occurred, there being produced only a "skim" containing from 15 to 20 per cent Sb, 65 to 60 per cent Pb, as mixed oxides. In the "fuming furnaces," where the temperature was only about 1400° F. and where an accumulation of "skims" from the primary softener was being treated with certain other antimony-containing materials, a considerable volatilization of antimony as oxide occurred; an "impure" skim of some 60 to 65 per cent Sb and 10 per cent Pb was also produced as mixed oxides. The latter material cannot be suitably treated in hearth-type or softener furnaces. The essential chemical difference between the first and second step is the customary use of some reducing material (such as petroleum coke) in the later operation.

An obvious interpretation of the effect of coke in facilitating the fuming of the antimony is the supposition that the antimony may be reduced to metal, and volatilized as such. This supposition is improbable, however, when it is realized that a slight extrapolation of the vapor-

* Published by permission of the Director, U. S. Bureau of Mines.

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[1] W. B. Hincke: The Vapor Pressures of Antimony Trioxide. *Jnl.* Amer. Chem. Soc. (1930) **52**, 3869.

pressure data for antimony metal[2] shows the vapor pressure of this material to be approximately 0.75 mm. at 1400° F. (760° C.), whereas the figure obtained by Hincke at the same temperature for the pure trioxide would approximate 23 mm. It seems certain that lead oxide is more readily reducible than antimony trioxide, so that actual conditions in the "fuming" furnace would favor a low concentration of antimony metal in the lead phase, and a high concentration of antimony trioxide in the oxide phase. Therefore the relative lowering of vapor pressure due to solubilities in each phase would increase the difference in the vapor pressures indicated. It must be concluded that the conditions in the two types of furnaces are truly anomalous with respect to volatilization, in that the antimony could not preferentially vaporize as metal.

It therefore became highly pertinent to study the melting points of mixtures of antimony and lead oxides, in order to determine the possible existence of compounds between the oxides that might cause this unexpected change of volatility, and to measure the vapor pressures of antimony trioxide above these mixtures.

The following report first presents the results of direct determinations of the melting point. Because of the corrosive nature of these melts, and because of other factors discussed below, many difficulties were encountered which prevented the direct measurements from furnishing completely conclusive results. Subsequently there are reported vapor-pressure measurements that were made on both solid and liquid mixtures, not only for immediate application of the data to the process of lead softening, but also because these measurements were of great value in fixing the constitution diagram. Microscopic examination of thin sections of various melts aided understanding of the system.

Direct Determinations of Melting Points

Melting points necessarily were determined in an atmosphere of nitrogen, because of the considerable speed of oxidation of both Sb_2O_3 and PbO to higher oxides, and of lead antimonites to lead antimonates upon heating in air. Suitable apparatus consisted of a large closed silica tube, mounted upright in an electric pot furnace and containing a suspended crucible with the sample and a thermocouple. The latter, as well as the nitrogen delivery tube, were brought in through a tightly fitting, heat-shielded rubber stopper. Platinum or silver crucibles were used for samples with lead oxide content of less than 75 per cent by weight. Neither platinum nor silver would withstand the corrosive action at the higher temperatures of fusion of the molten mixtures having

[2] O. Ruff and G. Bergdahl: Die Messung von Dampfspannungen bei sehr hohen Temperaturen, nebst einigen Beobachtungen über die Löslichkeit von Kohlenstoff in Metallen. *Ztsch. f. anorg. u. allgem. Chem.* (1919) **106**, 76.

high lead oxide concentration, but molten high lead oxide mixtures were viscous enough so that an alundum melting crucible could be used, providing the melts were made rapidly enough to prevent appreciable reaction between Al_2O_3 and the melt.

A platinum platinum-rhodium thermocouple was used for compositions up to 60 per cent PbO. Above this a base-metal couple withstood better the corrosive action of the molten sample. The thermocouples were standardized by comparison with a standard platinum couple calibrated by the Bureau of Standards. The thermocouple always was carefully centered in the finely powdered and well mixed sample and remained centered during fusion and solidification. All the melting points were determined by time-temperature curves on the cooling sample, care being taken, for reasons which will appear, not to heat much above the melting point. The amount of sample used was from 6 to 12 grams, depending upon the size of the crucible.

Table 1 shows the results of the melting-point determinations.

TABLE 1.—*Direct Determinations of Melting Points*

PbO, Wt. Per Cent	Sb_2O_3, Wt. Per Cent	First Fusion, Deg. C.	Eutectic Temperature, Deg. C.	Crucible Material	Thermocouple
0	100	654		Platinum	Pt, Pt-Rd
10	90	625	538	Platinum	Pt, Pt-Rd
20	80	555	Obscured	Platinum	Pt, Pt-Rd
25	75	550	540	Silver	Pt, Pt-Rd
30	70	560	Obscured	Silver	Pt, Pt-Rd
40	60	575	Obscured	Silver	Pt, Pt-Rd
50	50	574	Obscured	Silver	Pt, Pt-Rd
60	40	577	Obscured	Alundum	Base
69.5	30.5	575	Obscured	Silver	Pt, Pt-Rd
75	25	588	Obscured	Alundum	Base
80	20	652	621-End of Solidification	Alundum	Base
90	10	780	Obscured	Alundum	Base
100	0	890	Obscured	Alundum	Base

These results are better represented by the melting point diagram of Fig. 1. The region between 100 per cent Sb_2O_3 and 43.4 per cent PbO (the latter corresponding to the composition of a possible compound $PbO{\cdot}Sb_2O_3$) shows a normal melting-point curve with a eutectic at about 539, corresponding to 21.5 weight per cent PbO as well as specific melting points of pure Sb_2O_3 and the compound $PbO{\cdot}Sb_2O_3$ at 652° C. and 558° C., respectively.

In the succeeding region, from 40 to 75 per cent PbO, the observed melting points are nearly constant in magnitude, and were experimentally not well defined because of a low heat of fusion of the material. The

exact nature of the melts in this range obviously could not be deduced from the melting-point data, but vapor pressure and microscopic data discussed later furnish some evidence.

Above 75 per cent by weight PbO, the melting points of further samples, as indicated in the diagram, show a steep normal rise, increasing to the melting point of pure lead oxide, which was determined as 890° C. However, here again the heats of fusion are low and the experiments were necessarily made rapidly because of the danger of reaction of both the thermocouple and the crucible with the molten sample. Thus the

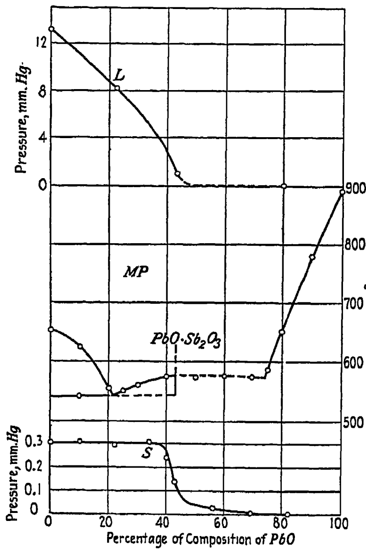

FIG. 1.

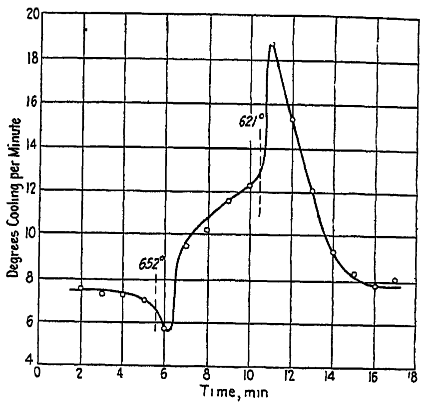

FIG. 2.

FIG. 1.—PERCENTAGE COMPOSITION OF PBO.
Curve MP is the melting point diagram.
Curve L is the vapor pressure of liquid mixtures at 697° C.
Curve S is the vapor pressure of solid mixtures at 539° C.
FIG. 2.—COOLING-RATE CURVE FOR 80 PER CENT PBO MIXTURE.
The higher temperature is the beginning and the lower the end of solidification.

melting points were best determined by plotting rates of cooling against time. The cooling-rate curve for a sample of 80 per cent PbO is shown in Fig. 2. This curve shows that freezing started at about 652° C. Ordinarily this would be taken to indicate that for this region of the curve PbO forms solid solutions with the antimonite, but another explanation of the behavior is indicated by microscopic evidence, and is discussed later. This experimental behavior was not completely reproduced by the determination of the 90 per cent PbO sample. The thermocouple broke on melting the sample the second time, which had been done in order to check the solidification range in a second experiment, performed after the temperature of first solidification was determined.

Vapor-pressure Measurements

The second part of these experiments involved the determination of the vapor pressures of Sb_2O_3 from solid samples of varying composition and from the liquid sample covering the same range. These vapor-pressure determinations were made at two temperatures. At 539° C. all the mixtures remained solid. Ten determinations of varying composition of solid material were made. The inert gas saturation method[3] was used and the samples were prepared from finely ground portions of mixtures previously fused in a silver crucible. Four vapor-pressure determinations were made at 697° C. on liquid samples fused in a silver boat and ranging in composition from 18 to 100 per cent Sb_2O_3. These two sets of experiments are shown in Table 2, and graphically as the upper and lower curves of Fig. 1.

Table 2.—*Vapor-pressure Measurements*

Solid Sample		Liquid Sample	
PbO, Per Cent	Vapor Pressure at 539° C., Mm. Hg	PbO, Per Cent	Vapor Pressure at 697° C., Mm. Hg
0	0.303	0	13.0
10	0.311	22	8.18
22	0.295	43	1.02
34	0.308	82	0.08
40	0.241		
43.4	0.152		
43.4	0.130		
56	0.03		
69	0.01		
82	0.00		

These vapor-pressure determinations indicate that crystals of pure Sb_2O_3 are present in the solid samples up to compositions approaching 40 per cent PbO, in that the vapor pressures of Sb_2O_3 from the solid samples in this region of compositions was substantially equal to that of pure Sb_2O_3. In no case was a deposit of any material observed in the weighing tube of the vapor-pressure apparatus other than pure cubic or orthorhombic crystals of Sb_2O_3; which indicated that neither the lead oxide nor the lead antimonite had any detectable vapor pressure at these temperatures. Above 43 per cent PbO the Sb_2O_3 vapor pressure of the solid samples at 539° C. falls off rapidly to practically nothing at 70 per cent PbO composition, thus indicating that no crystals of Sb_2O_3 are

[3] W. B. Hincke: *Op. cit.*

present in the solidified melts over this range and that the small amount of Sb_2O_3 liberated may be from a slight dissociation of the lead antimonites to give some vapor of Sb_2O_3.

The vapor-pressure curve for the liquid samples, as shown also in Fig. 1, indicates that Raoult's law holds approximately over the region 100 per cent Sb_2O_3, to 43.4 per cent PbO by weight, provided the assumption is made that the system here consists only of Sb_2O_3 and the compound. Even in the liquid, apparently there is a solution of Sb_2O_3 in a compound of this composition: $PbO \cdot Sb_2O_3$. Beyond this point the partial vapor pressures of Sb_2O_3 fall off to a very low value. At 82 per cent PbO the vapor pressure is just measurable by the methods used, while the solid material of the same composition had no detectable Sb_2O_3 vapor pressure.

Microscopic and Chemical Data

Thin sections were made of the various samples of which the melting points had been determined and were examined with a polarizing microscope.

In the range from 0 to 40 per cent PbO, no unusual characteristics were exhibited by these samples. Below the eutectic the solid system consisted of orthorhombic crystals of antimony trioxide with residual eutectic masses. Crystals of the compound $Sb_2O_3 \cdot PbO$ were well developed in the range rising from the eutectic, but there was some tendency towards a vitreous fracture. The crystal habit of the compound was similar to that of the pure Sb_2O_3, and the two materials were distinguishable with difficulty.

Beyond 43.4 per cent PbO, corresponding to the compound, progressive change was noticed. Samples were prepared corresponding to 53.4, 60.4, and 69.5 per cent PbO, representing the molecular proportion $2Sb_2O_3 \cdot 3PbO$, $Sb_2O_3 \cdot 2PbO$, and $Sb_2O_3 \cdot 3PbO$, respectively. Macroscopically these showed an increasing darkness of color and increasingly vitreous fracture, the last sample forming only a viscous tarry mass upon fusion. Under the microscope, the 53.4 per cent material showed crystals of $PbO \cdot Sb_2O_3$, and residual darker colored material which was isotropic, and probably of an amorphous nature. The composition 60.4 per cent PbO also showed crystals of $PbO \cdot Sb_2O_3$ in smaller amounts, and a considerable residue of the amorphous material. In the sample at 69.5 per cent PbO, two marked changes were apparent. The material was nearly completely optically inactive, and had only faintly distinguishable particles of the compound. The yellow translucent glass which represented the greater portion of the sample contained many minute opaque globules, uniformly distributed, which were finally shown to be inclusions of metallic lead. In the sample of 80 per cent PbO the inclusions of metallic lead were marked and easily recognizable.

Inasmuch as all samples used in this work were prepared and melted in an atmosphere of pure dry nitrogen, the presence of lead in these higher lead mixtures can be explained only through the reactions:

$$PbO + Sb_2O_3 = Pb + Sb_2O_4$$

or

$$2PbO + Sb_2O_3 = 2Pb + Sb_2O_5.$$

In this our results are contrary to those of Leroux[4] who was unable to find lead in fusions of PbO and Sb_2O_3 made in air. It is probable, however, that under the conditions of his experiments any lead which might have been first formed had opportunity to reoxidize. It is also apparent that our experiments have not confirmed the results of Tammann,[5] who found a compound corresponding to lead orthoantimonite, $Pb_3(SbO_3)_2$, formed upon heating equal molecular proportions of PbO and Sb_2O_3 as powders to 500° C. The compound was found as the residue upon extraction of the sintered mass with alcoholic acetic acid, and subsequently with dilute tartaric acid. It is not only possible but probable that the normal orthoantimonite was formed by ionic reactions upon dissolution and did not necessarily exist in the solid sample.

That lead antimonate is readily formed by the action of air on the antimonite was shown in an accessory experiment. Samples of composition corresponding to normal orthoantimonite, both before and after fusion in nitrogen, were heated in air just below and above the melting point of this composition mixture, and the increase in weight of sample was noted. This increase was exactly that necessary to increase the oxygen combined with 2 gram-atoms of antimony from 3/2 O_2, to 5/2 O_2. The black vitreous fusion when so treated formed a brown or brownish red amorphous solid, which resisted all attempts at fusion below 1100° C.

The oxidation of trivalent to pentavalent antimony by lead oxide in the melts from 70 to 90 per cent PbO offers explanation of the behavior observed in the cooling curve of Fig. 2. It is supposed that the initial solidification point approximately represents the melting point of the 80 per cent PbO mixture, but that in slow cooling the reaction accounts for the disappearance of some lead oxide, and an enrichment towards the antimony side of the diagram corresponding to a lower melting point, the antimonate formed being slightly soluble in the melt, and being responsible for its high viscosity. Therefore we have not shown this part of the diagram as indicating solid solutions but have drawn it as a simple melting curve. It is obvious that no great accuracy may be claimed for

[4] H. Leroux: Untersuchung über die Entfernung des Antimons aus dem Werkblei auf trockenem Wege. *Metall u. Erz* (1924) **18**, 54.

[5] G. Tammann: Chemische Reaktionen in pulverförmigen Gemengen zweier Kristallarten. *Ztsch. f. anorg. u. allgem. Chem.* (1925) **149**, 21.

the melting points in this region, since the composition of the melt was changing continuously. In carrying out direct melting-point determinations as rapidly as possible it seems probable that the limits of solid and liquid transitions have been defined, but no great definiteness can be assigned to the figures on this range as true melting points.

Application of Data to Lead Softening

The results obtained in these experiments seem definitely significant to an understanding of the general course of the process of lead softening. The formation of the compound $PbO \cdot Sb_2O_3$, present in both liquid and solid melts, clearly explains the anomalous behavior of antimony volatilization, and the oxidation of trivalent antimony at the expense of lead oxide in higher concentrations gives a possible picture of the mechanism of antimony removal from the bullion.

Consider first the vapor-pressure relationships appearing in "skims" or dross as actually produced in "hearth refining" furnaces. In the normal oxide mix first produced by the softening process, and containing approximately three parts of lead to one of antimony as oxide, the composition is located well to the right of the compound in Fig. 1, and there is no appreciable partial vapor pressure of antimony trioxide above the softener bath at temperatures of 1300° to 1400° F. Skim produced at this temperature necessarily must contain antimony in a state of oxidation above the trivalent form, whether the atmosphere be neutral or slightly oxidizing. When such material proceeds to the "fuming" furnace, the carbonaceous material must first reduce the lead content in preference to the antimony, since the former is the less stable oxide. The melt thus becomes enriched in antimony, and when the composition passes below 43.4 per cent PbO the vapor pressure of the antimony trioxide suddenly becomes appreciable and increases rapidly as further reduction proceeds. The subsequent behavior of the impure skim remaining shows definitely the great difficulty of reducing the portion of the skim that contains true lead antimonate. Our experiments show, by the failure to melt this material below 1100° C., that it must be a relatively stable and inert substance, and it seems likely that this material should not be reduced by the carbonaceous material.

It may be supposed, however, that the higher oxides of antimony, or lead antimonates, which probably have limited solubility in the various "skims," are the media responsible, at least in part, for the oxidation of the metallic antimony of the bullion; that is to say, they may act as oxygen carriers or catalysts. Doubtless a steady or balanced condition exists in actual practice where the dual method of refining is used. The higher oxides of the bath tend to be reduced by the antimony of the bullion, but the balance is maintained by the opposing tendency of the

lead oxide, and whatever air may be present in the furnace gases, to reoxidize the trivalent antimony oxide so formed. Should an excess of the latter condition prevail, lead will be oxidized, and the skims will carry too much of this material; but should the balance tend too far to the former condition, the softening process will be much retarded.

From the point of view of practical operation, the formation of excessive amounts of lead antimonate is to be avoided, but it will be apparent from the above that some content of antimony in the higher state of oxidation is unavoidable, and may be essential for the successful prosecution of the softening process. Even though the softening should be carried out by litharge alone, as is sometimes done, higher oxides of antimony would still be produced in an inert atmosphere.

The actual process of lead softening is much more complicated than the simple and more or less "static" factors described above. In practice, the usual method of softening involves at least two major factors capable of improvement and worthy of investigative study. The relative slowness of operation and the often erratic operation of the hearth type of softener for lead cause considerable amounts of lead metal to be sequestered about the plant, and the high lead content of the skims or scoria apparently unavoidable at present also causes a considerable circulating load of oxides and further accumulation of metal as oxide in the plant. The process as carried out in hearth furnaces is capable of but little variation in practice. The authors know of several instances in which young and enthusiastic metallurgists have sorrowfully found that even slight variations of furnace conditions, aimed at bettering the rate of softening or lowering the lead content of skims, disastrously affected the operation.

Before the data supplied above can be effectively used in studying the process of hearth softening, or lead to improve methods of softening, two important items remain to be investigated.

The actual process of softening represents some sort of "steady state" involving *rates* of oxidation of bullion and slag. A prerequisite to definite understanding of the hearth type of lead-softening process is knowledge as to the relative rates of oxidation; the relation is shown as follows:

1. Bullion
   - (*a*) Sb by elementary oxygen
   - (*b*) Pb by elementary oxygen
   - (*c*) Sb by higher oxides of Sb or Pb
   - (*d*) Pb by higher oxides of Sb or Pb

2. Slag
   - (*a*) Lower to higher oxides of Sb by elementary oxygen
   - (*b*) Lower to higher oxides of Sb by lead oxide

Furthermore, in order to interpret such data, the chemical stability and the reducibility by reducing gases of the various oxides involved must be determined.

The Pacific Experiment Station of the Bureau started some rate measurements similar to these outlined above some time ago. The technique of such measurements in the laboratory is difficult and involved, and these difficulties had scarcely been solved when unavoidable change of personnel caused the temporary suspension of the experiments. In the interim, a start had been made in the study of reducibility by entropy determinations on the metals and oxides of the fifth periodic group.[6]

The very considerable possibilities in the improvement of present softening processes and the gradual accumulation of fundamental data which may ultimately be used for this purpose justify the continuance of these experiments, either at this laboratory or elsewhere. The publication of the results, which have been held for some time in hope of being enabled to present a more complete story, seems desirable to avoid possible repetition, and to assist in a possible analysis of the more practical problem of improving present methods of softening.

Summary

The melting-point diagram of the system $PbO-Sb_2O_3$ has been determined.

The compound $PbO.Sb_2O_3$, corresponding to 43.4 per cent PbO, exists in both the solid and liquid state.

Compositions below 43.4 per cent PbO show a simple eutectic diagram, with a minimum melting point of 539° C. at 21.5 per cent PbO.

An ill-defined second eutectic probably exists at approximately 74 per cent PbO, and melting takes place at 575° C.

Compositions above 70 per cent PbO show an oxidation of Sb_2O_3 to higher oxide, with the formation of metallic lead, and the melts become vitreous in this range.

Vapor pressures of solid and liquid mixtures have been determined at 539° and 697° C., which show that antimony trioxide is volatilized rapidly only at compositions with lead content below that of the compound $PbO\cdot Sb_2O_3$.

The behavior of the $PbO-Sb_2O_3$ system on melting has been discussed in an elementary way in relationship to practical processes of lead softening.

[6] C. T. Anderson: I. The Heat Capacities of Arsenic, Arsenic Trioxide, and Arsenic Pentoxide at Low Temperatures. *Jnl.* Amer. Chem. Soc. (1930) **52,** 2296; II. The Heat Capacities at Low Temperatures of Antimony, Antimony Trioxide, Antimony Tetroxide, and Antimony Pentoxide. *Ibid.,* 2712; III. The Heat Capacities of Bismuth and Bismuth Trioxide at Low Temperatures. *Ibid.,* 2720.

DISCUSSION

(Carle R. Hayward presiding)

C. R. Hayward, Cambridge, Mass.—I consider this to be a paper of fundamental importance in lead refining. I am in hearty agreement with all of the findings of the authors. Anyone who has experimented with the softening of lead and has seen the lead-antimony slag or dross, whatever it is called, that appears on the surface during that operation, particularly in the usual method of smelting battery plates without flux in reverberatory furnaces, can readily see that the findings here explain many of the phenomena. A number of years ago in experimenting with the removal of antimony from lead by agitating it with litharge, we produced at temperatures between 600° and 800° C. this very liquid product on the surface, and I was rather interested to find out its actual melting point. Naturally, as the authors have pointed out, it is difficult to determine this with thermocouples without serious corrosion, so I thought it would be an easy thing to cut some of it into the form of cones and melt it under optical observation. I took some cones and gradually heated them in a muffle furnace, but the temperature continued to rise and the cones refused to melt until far above the original melting point. I was much puzzled by this behavior but made no effort to find the cause. It is evident from the authors' work that the antimony was originally present as Sb_2O_3 but when the cones were heated it was changed to Sb_2O_5.

E. L. Jorgensen, Irvington, N. J.—The theory is advanced that the higher oxides of antimony and lead antimonates act as oxygen carriers or catalysts. This theory is confirmed in another field. When oxidizing lead to red lead for pigment, an addition of 1 lb. of antimony for 3000 lb. of red lead will reduce the time of oxidation by about 15 per cent.

Quarry Waste in the Indiana Limestone District

By J. B. Newsom,* Bloomington, Ind.

(New York Meeting, February, 1932)

In the Indiana limestone district, some 50 or 60 per cent of the merchantable stone in a quarry opening is waste, and only about 40 or 50 per cent of the stone from the opening is finally sold. So long as the present system of quarrying is used, the wastes measured and reported in this paper will continue. Comprehensive development of an entirely different method of quarrying, using wire saws for cutting, promises great improvement. Some of the savings to be expected are pointed out in the following pages. No attempt has been made to measure losses due to variations and natural flaws in the stone, since these vary greatly in different quarries, and accurate measurement would be difficult. The wastes that have been measured account for about 30 per cent of the total ledge, which leaves 20 or 30 per cent chargeable to unmeasured sources of loss.

General Causes of Waste

There are four general reasons for quarry waste: (1) the structure of the deposit, such as its shape, bedding planes, solution cavities and strain fractures; (2) efforts to quarry blocks which do not contain distinct color or textural variations; (3) the trade custom which permits purchasers to specify the size of the quarry blocks which they will buy, which is a matter of competition for business on the part of stone producers; (4) quarry methods, which cause the greatest amount of waste. Wide channeler cuts, hook holes, crooked splitting and uneven breaking on quarry floors are included in this category.

Waste Due to Structure of Deposits

The peculiarities of structure cannot be eliminated, but a chance for saving lies in quarrying to take advantage of them.

The commercial deposits of Indiana limestone are lenticular masses from 20 to 70 ft. thick, and nearly horizontal. All of these deposits of good stone are in the Salem limestone, a bed in the sub-Carboniferous. The Salem is traversed by two systems of vertical fractures, a major system running from northeast to southwest and a minor system running from northwest to southeast. Where the stone is covered by an over-

* Engineer, Bloomington Limestone Co.

burden of fairly dense limestone or a bed of shale, the fractures may cause little trouble; where there is no protective covering, solution along the fractures may have formed irregular cavities known as "mud seams."

Where mud seams are numerous it is best to quarry with the main cutting lines parallel to the main seams, for two reasons. In the first place, changes in color work outward into the surrounding rock from the mud seams. If the quarry is operated along the seams the chance of a block running through a change in color will be less. The quarry will not produce so many "variegated" blocks. In the second place, if the mud seams are close together, a quarry operated across them will necessarily produce many short blocks which cannot be used. If the quarry is operated parallel to these seams, they form the back of many blocks which, although they may be irregular in shape, can be used.

Strain fractures, called "dries" by quarry operators, usually run more or less parallel to the mud seams, though they may run in any direction. They are a constant source of waste which seems difficult to eliminate, as often they are not seen until after the cutting has been done. Possibly if dries were marked on the top of each floor as the floor above was being removed, some of this waste could be avoided. At present operators trust to memory for the location of dries which extend from the upper floors downward.

Some stylolites do not interfere with the strength of the stone, while others form lines of discoloration and weakness which often cause the loss of a bed of stone 1 ft. or more thick. As quarries are operated, the stone is removed in benches about 10 ft. thick. When it is possible, operators plan the benches so that stylolites come either at a bench floor line or midway between two floor lines. Unfortunately, a stylolite is seldom level, so it may coincide with the floor line at one place and run above or below it several feet in another place, with a resulting waste of the stone between the stylolite and the quarry floor.

It is necessary to carry level floors when operating with channelers, as they will not operate to advantage on sloping tracks. If wire saws are used and channelers eliminated, level floors will be unnecessary, so the floor breaks often can be made to coincide with the stylolites.

Waste Due to Demand for Blocks of Uniform Texture

Many stylolites, calcite seams, and such flaws do not result in structural weakness or appreciably increase the difficulty of working. However, the market will not absorb all the variegated stone produced, so the quarryman must eliminate these textural lines and variations. This is done by spalling or splitting the blocks till they are so small that only clear stone remains. Heavy waste is the result of this practice.

Fortunately for the stone trade, architects are more and more interested in variegated wall surfaces, and are building up a demand for stone

that varies in texture and color. A good example of this tendency is found in the growing demand for stone (classed as "Old Gothic" by the trade) full of large calcite spots, small solution cavities and other marks. Another example is found in the desire for uneven color in stone to be used in rock face walls. This demand has become so insistent that sometimes iron filings are spread over the stone, so that rust from the filings may stain the stone and produce color variation.

Trade Competition Which Causes Waste

Each stone job differs from every other in the size and shape of the face blocks, the size and shape of the moldings, the columns, the carvings, and other features of the stone work. Retailers all over the country must be prepared to cater to these differing demands, therefore they want quarry blocks of special sizes for each job, in order to cut down mill waste. Many orders specify the exact size of the quarry blocks to be supplied, taking no account of the blocks that happen to be in stock.

Mill equipment also may demand certain block sizes. One mill may have extra long saws which need long narrow blocks for best sawing efficiency, while another mill may have short wide saw beds that work best on short blocks.

In order to fill the retailer's size specifications, many blocks are measured short, trimmed, or one end split off, resulting in a heavy waste to the operator. Probably the best solution for this difficulty lies in the pricing method already in use, by which one price is charged for promiscuous sizes, and another for specified sizes. This scheme is open to the objection that it fails in times of keen competition for business. An attempt to standardize block sizes probably would not be successful because variation in size is a natural result of the flaws and texture changes found in every ledge.

Waste Due to Method of Quarrying

At present, quarries are operated by channelers served by 30-ton derricks. The derrick size places a definite limit on the weight of a block that may be removed from the quarry. Railroad flat cars are 8 ft. 6 in. or 9 ft. wide. Gang saws are often built to take blocks not over 12 ft. long by 6 ft. square. Since waste is in proportion to the surface area produced or the number of blocks quarried, such limits on block size have definite effects. A quarry producing mostly small blocks is sure to have a high ratio of waste.

Turning to the actual methods of quarrying now in use and ignoring the waste caused by natural flaws, we have to consider losses due to: (1) stone cut up by channelers; (2) channelers not cutting straight; (3) bottom breaks; (4) splitting channeler cuts into quarry blocks; (5) hook holes; (6) the system of measuring stone.

In order to estimate properly the loss due to each of these six sources, careful measurements and averages were taken. To be sure of accurate results, the observations were made at two quarries, one of which is considered to show little loss in quarrying and the other to be about average. Fifty of the 100 measurements from which an average was made were taken at each quarry. No attempt was made to select blocks when the measuring was done.

The losses measured are lower than the actual losses in the quarries, because some blocks are so badly split or cuts so shattered in turning that the stone is thrown directly on the waste heap and such blocks escaped measurement.

Loss Due to Channeler Cuts

Electric channelers are used in most quarries. These operate on a track which must be set nearly level, so that the machine can travel back and forth without being moved appreciably by the jar of the strokes. Modern channelers carry two sets of drills, one on each side of the machine, with a distance from center to center of the drill gangs of 8 ft. 5¼ in. Each gang cuts out a channel 1¾ in. wide. The usual method of quarrying is to cut on both sides of the machine and then to move over half a channeler width. This results in slicing the ledge into sections 48⅞ in. wide, with a 1¾-in. slot between slices. These sections or slices of a ledge are called "cuts" by the quarrymen. In addition to this cutting it is necessary to free completely both ends of each slice before an attempt is made to remove it. For this purpose cross channels are put in about every 55 feet.

After the channeler cuts are made, each slice of stone is wedged loose on the bottom, pulled over so that it lies flat, split into blocks small enough for the derrick to handle and lifted out of the way. To make this operation possible it is necessary to quarry the stone from one side of the floor in order to provide a place to turn down the first cut. This is done by taking out what is known as a "key block row." In quarrying a key block row, channelers do about one and one-half times as much cutting as on the other rows.

Calculation of the loss due to pulverization of the stone by the channelers gives 4.13 per cent of the total floor. Some operators use wider bits, which materially increase the above percentage.

Assuming that the present method of quarrying is sound, there seem to be two possible lines of attack to reduce the loss due to channeler cuts. If each cut were widened to say 5, 6, or even 8 ft., the effect of this particular loss would be lessened. However, such a procedure would greatly increase the loss due to splitting up channeled sections after they are pulled over on their sides. (See p. 113.) Measurements indicate that this increased splitting loss would more than offset the gain to be effected

by taking wider cuts. Another possible remedy lies in using narrower drill steel. In order to cut rapidly a channeler drill must be fairly rigid and narrower steel would not have this feature. Also, channeler cuts tend to run crooked, causing loss of stone, and narrower steel probably would increase this tendency.

Wire saws would almost entirely eliminate this source of waste, as a wire saw cut is only about 1/4 in. wide. If wire saws were used, therefore, this loss would be cut from 4.13 to about 0.6 per cent.

Loss Due to Crooked Cuts

Channeler steel often strikes variations in the stone, which cause it to run crooked. Loss of stone due to this effect is rather smaller than would appear, because the cuts are split into small pieces before measuring. To illustrate: If a gang of channeler bits runs 4 in. in the 10-ft. depth of a floor, so that the cut as turned down is 4 in. narrower at the top than at the bottom, there will not be a 4-in. loss of stone, because two or three blocks will be cut from the 10-ft. height. If three blocks are cut, the loss on each block will be 1 1/3 inches.

In order to estimate the stone lost from this cause, opposite ends of 100 blocks of stone were measured. The average variation was 2.05 in., which equals 5.02 per cent of the stone left after the channelers have finished cutting.

Some channeler gangs are built up of five separate bits each 1 by 1 1/2 in. in cross-section. Other gangs consist of three bits, each 1 by 2 1/2 in. As each type of gang is 1 in. thick, they should offer about the same resistance against running sideways. A stiffer steel or a different method of sharpening channeler steel might bring results. Thicker drill steel does not seem a feasible solution, as this would necessitate heavier channelers and appreciably raise both the first cost of the machinery and the operating cost.

Therefore, if channelers are used, this cause of waste probably cannot be eliminated. A change to wire saws would be an improvement, as a wire saw can be made to cut within 1/4 in. of a true vertical surface.

Loss Due to Bottom Breaks

Channelers are designed to make cuts about 10 ft. deep, so when a floor has been cut to that depth, it is necessary to remove the stone before the machines can be put on the next floor below. Operators try to cut 10-ft. floors, but if there is a horizontal flaw in the stone near the 10-ft. depth, they bottom their drills in the flaw, making a part of the floor waste coincide with the flawed stone.

A floor must be horizontal, for the sake of economy in leveling the channeler tracks, therefore each cut must be split off low enough so that a level floor results. This splitting off is begun by clearing away the

stone from the side of the cut to be removed. Then holes are drilled diagonally downward from the bottom of the cleared side. These holes are placed about 6 in. apart and are 6 in. deep. Wedges called "plugs" are placed in the holes, between two pieces of half-round iron called "feathers." Tapping on the plugs with a hammer will start a split along the line of holes. Further tapping will extend the fracture until it turns up and runs to the bottom of the next channeler cut. The holes are drilled at a downward angle so that the fracture line will have to turn up to reach the bottom of the channeler cut and because, being placed in a corner, they cannot be drilled horizontally. The result of this procedure is a rounded bottom on each cut, the bulge of the rounded part representing lost stone. The average bulge is 7.45 in., which represents a loss of 6.58 per cent of the stone left at this point. This is much below the true figure, for many blocks split far above the expected line and are thrown away.

Almost invariably a split such as has been described follows the drill holes to the bottom of the point where the plugs and feathers exert pressure. After leaving this line, the break follows a curved path. If it were unnecessary to carry level floors, it would seem feasible to drill holes nearly through the cut and to put in long plugs and feathers in order to carry the strain lines straight through. Apparently this has never been tried carefully. It is not feasible so long as level floors are necessary. If the holes were put up 3 or 4 in. and drilled down at a slight angle, as is common practice in marble quarries, there would be a saving in stone; but it might be so slight as hardly to pay for the extra cost of drilling.

The loss due to bottom breaks is dependent on the distance between channeler cuts, regardless of the distance between floors. If higher benches were carried, say 20 ft., this loss would be materially lowered. This has been tried by the Shawnee Stone Co.; but it was found that the extra weight of the long drill steel, the difficulty of freeing stuck drills, and the difficulty of turning over the cuts without shattering the stone, more than offset the gain.

Wire saws probably would greatly lessen this loss, because they would make it unnecessary to carry level floors. It would be possible to break to a flaw or to break successive cuts higher, drilling them straight through, as explained above.

Loss Due to Splitting Cuts into Blocks

A cut, as turned down, weighs about 150 tons, and must be split into smaller blocks before it is removed from the quarry. This splitting is done as follows: The ledge foreman examines the cut after it has been pulled over on its side, noting flaws and variations in texture, and marks

it into a series of rectangles representing finished quarry blocks. Six-inch air drill holes are put down at 6-in. intervals along the lines marked out and plugs and feathers are put in the holes, after which the blocks are split by tapping, as described. In this case, after the split leaves the line of drill holes, it follows the natural bedding planes or rift of the stone to the other side of the cut. If the stone is cross-bedded, a heavy waste will result from the tendency of the split to follow the rift. The average run from this cause is 6.58 in., which equals 12.33 per cent of the stone remaining at this point in the operation.

Obviously, if larger blocks could be handled the waste due to crooked splitting would be lessened. The present tendency toward heavier hoisting equipment is a step in the right direction. In some sandstone and marble quarries loss due to splitting crooked is cut down by wedging all the way through the blocks. Quarrymen here say this procedure would not be as successful as the present method on Indiana limestone, because the extra stone would not be worth the trouble.

Waste Due to Hook Holes

Blocks are handled by large hooks, known as "dogs," which are fastened to chains in such a way that the weight of the block will make the dogs grip more tightly. Dogs are apt to break out unless they are securely seated against the stone, therefore a seat, or "dog hole," is picked in both ends of each block. As stone is sold on the basis of the largest perfect rectangular block, measurements must be shortened to allow for dog holes. This takes 2 in. off the selling length of every block and 4 in. off blocks which have straight end splits. The purchaser will recover the length in most of the blocks, but from the producer's standpoint this is a serious loss. It tends to make the milling loss seem smaller than it really is, and the quarry loss larger.

Probably no tackle could be devised which would be as satisfactory as dogs for handling stone in a quarry. For small pieces of stone, clamps arranged so that the weight of the stone would make the clamp grip tightly probably are feasible, but the blocks in a quarry are of many sizes and shapes and it would be difficult to design a clamp that would adjust itself to these inequalities.

Loss Due to System of Measuring

It is usual to allow a purchaser 1 in. on every dimension of a block for the certainty of giving full measure. As with the dog-hole loss, the main objection to this practice is found in the fact that it throws an extra apparent loss on the quarry and tends to reduce the apparent milling loss. On the average mill block, the measuring loss decreases the selling size by about 5.29 per cent. If blocks can be produced which are true

rectangles, this allowance will no longer be made because the stone producer will be sure of his sizes.

Summary of Data

Considered separately, the figures given tend to be confusing. A better appreciation of their significance will be gained from a study of Table 1, which is constructed by working from the quarry production, although this makes the figures lower than they would be if all the sources of waste could be measured and included. To illustrate, the channeler loss indicated applies only to the stone actually produced. The channeler loss incurred in producing stone which was damaged in some later process, such as in splitting cuts down to quarry block size, is not shown. If the total quarry loss is assumed to average 50 per cent, channeler loss should be 82,600 ft. instead of the 59,450 ft. shown. The other losses also would be proportionately larger. However, these figures are accurate enough

TABLE 1.—*Measured Quarry Waste*[a]

	Cubic Feet
Cut by channelers	59,450
Lost by crooked channels	69,270
Bottom-break loss	86,200
Lost splitting to mill-block size	151,050
Dog-hole loss	17,840
Loss due to measuring short	55,950
Finished stone	1,000,000

Total measured loss, 439,760 cu. ft. out of a total of 1,439,760 cu. ft., or 30.5. per cent.

[a] Figures apply only to the waste on the stone actually produced. Work done on stone spoiled in process is not included in this table.

to indicate which causes of quarry loss are at present the most serious. Applying them to the 1930 production, we find a total loss of 4,200,000 cu. ft. worth about $2,050,000.[1]

Wire saws have not yet been used in Indiana to a sufficient extent to fully capitalize their advantages. When they are so applied, the savings described will be realized. This will mean that the loss due to the method of quarrying will be cut to about 20 per cent instead of the 30.5 per cent shown in the table. The stone thus saved would have been worth $720,000 in 1930.

ACKNOWLEDGMENT

The author's thanks are due Mr. W. H. Johnson, president of the Bloomington Limestone Co., for permission to publish the figures contained in this paper.

[1] Value of rough stone from Department of Commerce statement No. 9632, released June 9, 1931.

DISCUSSION

(*W. M. Myers presiding*)

J. R. THOENEN, Washington, D. C.—Is there any waste due to the jarring of the strata on either side of the channeler cut?

J. B. NEWSOM.—No, because the limestone is resilient and takes up the jar. Where cuts meet at right angles, at the ends, however, there is some spalling off. This is not important, as the end lines are put in only every 60 feet.

Results of Wire Saw Tests

By J. B. Newsom,* Bloomington, Ind.

(New York Meeting, February, 1932)

During July and August, 1931, the Bloomington Limestone Co. at Bloomington, Ind., ran a single wire saw on ledge No. 2 at Maple Hil quarry. The ledge was hard rock, much harder than the average Indiana limestone. It was opened at two ends for a depth of about 20 ft., leaving a 130-ft. section between. During the tests 21 cuts were made in this ledge, to an average depth of 10.42 ft., for a total effective cutting area of 28,445 sq. ft. The actual area sawed was about 20 per cent higher, as we did not level down in the center, preferring to cut deeper on the ends

The various cuts were carefully logged, and the results plotted. A study of these results is interesting, as it enables a forecast of probable future costs to be made, and also because it points out the limitations of wire saws, and the lines development should logically take. Some of the inferences to be drawn are discussed below.

Best Length of Cut.—No trouble was due to length of cut, except that we could not easily level down a long cut, and it is probable that longer cuts, say up to 200 ft., can be made. Obviously, longer cuts will result in a higher sawing rate in square feet per hour if they do not cause stoppages.

Working Higher Ledges.—The cuts averaged 10.42 ft. high by 130 ft. long. A study of several logs indicated that during only about 15 per cent of the cutting time was the saw cutting to its full efficiency; that is, during about 85 per cent of the time the saw was bedding in at the ends and not cutting well in the center, or the ends were below the floor line and only part of the wire was doing effective cutting. This does not mean that the wire was cutting to only 15 per cent of its capacity; but it means that during about 85 per cent of the time part of the cutting was being wasted.

If higher ledges had been worked, the wire would have cut to its full efficiency a large part of the time, as it would have had a greater distance to saw after it had bedded in to the ledge before the ends went down below the floor level. The exact saving to be expected cannot be calculated, but it would be large if really high cuts were made.

* Engineer, Bloomington Limestone Co.

Leveling Bottoms of Cuts.—On the first cut, which was 150 ft. long, the bottom was leveled to within about 4 in. However, it was a slow process, and made the cost unusually high. If several cuts had been running at the same time, so that the leveling down would not have delayed other work but would have taken only power, water and sand, the cost would have been lowered considerably.

On all except the first cut, floor level was reached by cutting below the level on the two ends. Measurement of eight bottom profiles indicated that it is possible to come within the following limits without appreciably slowing up the work:

Length of Cut, Ft.	Lag in Center, Ft. and In.	Length of Cut, Ft.	Lag in Center, Ft. and In.
25	0′ 6″	100	3′ 0″
50	1′ 0″	125	4′ 6″
75	2′ 0″	140	6′ 0″

Sawing Down in Old Cuts.—We broke a wire near the bottom of a cut, and threaded in a new wire, sawing down from the top in the old cut. There was trouble, as we could not hold sand on the wire, and had to saw through small pieces of flint and spalls all the way down. However, the wire did go down in about one-third of the time it would have taken to make an entirely new cut.

Removing Wire by Raising it Out of Cut.—This was tried on one wire which was not appreciably worn, with the idea of using it in another cut. As it proved much more difficult than we had anticipated, it was concluded that the best way to remove a wire is to cut it.

Splicing New Wire into a Partly Worn One.—A number of times a new wire was spliced into an old one, and it was found to be a perfectly feasible operation. We spliced in short lengths only, but without doubt a whole new wire could just as easily have been spliced in.

Best Size of Wire.—Two cuts were put down with $3/16$-in. wire. This wire sawed much faster than the $1/4$-in. wire while it was new, but wore out before the cuts were bottomed. Apparently the larger wire is the more economical, but the one test made was not enough to be conclusive. The small wire is certainly more expensive because it wears out so quickly, but it will save sand, power, wire tension and wear on equipment. Also, it will save labor, as a small wire is easily handled.

Double-twist Wire.—The cuts put down with single-twist wire all had a slight run to the east, not enough to interfere seriously with the shape of blocks but quite enough to make it difficult to predict the exact size of the blocks. Double-twist wire was used in two cuts. There was no trouble with it after we learned how to splice it, and it was found to make an absolutely straight cut. It might be wise to use it for cuts where close dimension work is desired.

Reclaiming Sand.—During part of the time we used reclaimed sand. The men grumbled at this, thinking that it did not cut as well as the new sand. However, measurements of areas cut were taken at the end of 8-hr. periods; during several shifts new sand was used and during several other shifts reclaimed sand was used. The reclaimed sand cut more per hour than the new sand. This may, of course, have been an accident of measurement or sawing.

Small Single Saws Versus Large Double Saws.—Several single saws seem to be a better arrangement than a few large saws. The sawing time on the last 10 single and double cuts was as follows:

	Total Hours	Net Hours	Percentage Effective
5 double cuts	318:45	227:25	71.35
5 single cuts	239:40	189:10	78.90

In other words, single-cut saws can be kept sawing a greater percentage of the time than double-cut saws.

To this saving in sawing time should be added the saving due to being able to move with a small labor force if small saws are used, the advantage of being able to move a saw each day if several saws are used, and the advantage of always working the moving crew during the daylight hours. These savings are partly counterbalanced by increased friction and motor losses if several separate units are used.

Value of an Experienced Crew.—As the crew gained experience, the sawing rate constantly increased. On the last double cut the cutting rate per sawing hour was 87 sq. ft. as compared with 21 sq. ft. on the first single cut on the floor. Part of this improvement was due to the fact that the cut was higher, so that less time was lost bedding in the saw and cutting below the floor level.

Cost Data Obtained, and Probable Future Costs

All the prime costs, including depreciation on equipment, were carefully kept, with the idea of determining what can be expected from wire saws. The actual figures obtained and forecasts of the possible improvement in each of the separate items of cost are as follows:

	Cents per Sq. Ft.
July cost	20.71
August cost	11.22
Total average cost	13.94

	Distribution of Average Cost, Cents per Sq. Ft
Labor	5.89
Sand	2.07
Wire	3.09
Repairs	1.26
Miscellaneous	0.93
Power	0.35
Depreciation	0.35
Total	13.94

Possible Maximum Sawing Rate on Same Grade of Stone.—From the records, it is possible to select periods when the saw was doing straight sawing, as distinguished from bedding in or leveling down. During a total of 61.50 such hours, taken from the last four cuts, the saw cut 3199 sq. ft., an average of 52 sq. ft. per hour. At this rate, the saw rigged double should cut 104 sq. ft. per hour, or four single saws, which would require about the same amount of labor, should cut 208 ft. per hour, on similar stone. From this should be subtracted 20 per cent allowance for moving, bedding in, repairs, etc., thus: 208 × 80 per cent = 166 sq. ft. per crew hour.

Consideration of Separate Cost Items

Labor.—From the data above, it seems reasonable to suppose that a sawing rate of 166 sq. ft. per hour can be maintained. At this rate, the labor charge would be 1.108¢ per square foot.

Sand.—On a ledge laid out for proper drainage, the sand could be used until discarded by a classifier. Also, on a high ledge, the direct loss of 20 per cent due to cutting below floor level would be greatly reduced. It is impossible at present to figure accurately the saving possible if both these factors are considered. However, it seems probable that 50 per cent reduction in this item may be effected. At this rate, the sand charge would be 1.04¢ per square foot.

Wire.—The wire consumption constantly grew less, owing to the fact that the safety factor in wire life was being cut down. On the last cut, 3500 ft. of wire made 3434 sq. ft. of effective cut. Wire costs $19.00 per 1000 ft., or 1.9¢ per foot. If the wire is good for a square foot of cut to the running foot of wire, the cost should be 1.90¢ per square foot.

Repairs.—The repair bill was high (1.26¢ per sq. ft.) because we were just feeling our way and were working in a place not well suited to wire saws. Repairs should not cost more than 1¢ per square foot.

Other Charges.—These items, totaling 1.63¢ per foot, will change little unless softer stone is cut.

Improvement to Expect.—Adding the items listed above, we find that operating the saws properly, on similar hard stone, we should be able to cut the costs to about 6¾¢. On ordinary stone, it should be possible to cut this in half again, thus finally coming to a cutting cost of about 3½¢ per square foot.

After the wire-saw experiments were finished, the second floor of the same stone was worked with channelers to gain a cost comparison. This comparsion is not entirely fair, because the channelers were fully developed machines, in the hands of an expert crew, operating on a ledge with both ends open for drainage, and ideal in every way for operation of a channeling machine. Also, the machine men felt that their jobs depended on making speed, as they knew that the work was being checked against

the previous work of the wire saw. Under these conditions, the channeling prime cost was 15¢ per square foot cut.

As mentioned before, the wire saw was at a disadvantage in being used on a shallow ledge, and in being in the hands of an inexperienced crew. Also, there were only one-half enough wire saws to keep the men occupied. In spite of these handicaps, the wire saw cut for 11.22¢ per square foot, as against 15¢ per square foot by the channeling machines.

From the analysis of costs, it seems safe to say that wire saws, properly operated, will cut for about about one-half the cost of channelers. As channeler cost usually is figured at about 8¢ per cubic foot of stone produced, the savings should be about 4¢ per foot. Applying this figure to the 1930 production, as reported in Department of Commerce Statement No. 9632, released June 9, 1931, we arrive at a possible saving during 1930 of about $550,000 in quarrying costs. In addition to this, successful application of wire saws should result in a substantial reduction in waste, which, when applied to the 1930 production and value figures, would have been worth some $720,000 in that year. Thus the total saving to be expected from successful application of wire saws in Indiana is well over one million dollars per year.

Acknowledgment

Thanks are due to the Bloomington Limestone Co. for permission to publish its cost figures on wire sawing.

DISCUSSION

(W. M. Weigel presiding)

J. R. Thoenen, Washington, D. C.—Is there any relation between wire footage used and square feet cut?

J. B. Newsom.—Yes. In the "hard top" material we count on 1 sq. ft. of cut per foot of wire, the wire being ¼-in. size.

S. H. Dolbear, New York, N. Y.—Is the wire actually consumed?

J. B. Newsom.—It is worn down until the grooves no longer catch the sand and drag it along in the cut. The wire is twisted strand, so the grooves between the strands drag the sand in.

S. H. Dolbear.—Then the wire is really used up?

J. B. Newsom.—It is worn out and thrown away. As an illustration of what the wire will saw through, I can tell one incident that occurred during the work. As this was in the way of a competitive test with channelers, one of the channeler crew (we suppose), put a piece of tool steel down one of the fissures in the rock, and the wire saw cut right through it. We discovered the end of the steel in the fissure after the cut block had been removed.

J. R. Thoenen.—What size sand do you use?

J. B. Newsom.—We employ −14-mesh sand, usually the rounded Ottawa sand. On granite we use shot.

A. R. Chambers, New Glasgow, N. S.—What size sand is rejected after being used?

J. B. Newsom.—The used sand is put through Dorr classifiers over and over again, until it is finally rejected when it is about −35 mesh.

J. R. Thoenen.—We have found that a fine grained sand is best for cutting slate.

Magnetic Beneficiation of Nonmetallics

By Samuel Gibson Frantz,* Princeton, N. J., and G. W. Jarman, Jr,† New York, N. Y.

(New York Meeting, February, 1932)

The purpose of this paper is to relate briefly the development of magnetic separation and its extension from the separation of iron into its present use in the nonmetallic field, to suggest possible future extensions of its field of usefulness and its possible cooperation with other separating methods. The engineering design of separating machines, and statistics as to the number of machines in operation, types, their location and production are outside the scope of this paper. Specific data will be given, however, on certain selected applications.

A large part of human endeavor consists in separating one thing from another; the good from the bad, the desired from the undesired. In fact, a philosopher might plausibly claim that separating things into categories and choosing between them is the object of thought itself. In dealing with matter, separation is nowhere more inherently necessary to attain our objects than in mining and metallurgy.

Both our desire and our ability to make separations are results of the physical differences between things. Gross separations such as removing ore from worthless rock or coal from slate are usually made visually and depend upon form and color, which are physical properties. When the desired and the undesired things are intimately mixed, however, it becomes impractical to select visually even if the individual particles are readily recognizable to the eye, and some automatic way of making the particles separate themselves on the basis of their own inherent differences in physical properties must be resorted to. Some of the physical properties which have been used as a basis of separation are: color, shape, hardness, size, density, elasticity, surface properties (surface tension in contact with liquids, etc.), electrical conductivity, dielectric constant, and magnetic susceptibility.

Differences in these properties are used in the following methods of separation: color sorting, sifting, settlement in air and in liquids, float-sink methods, flotation, electrostatic and magnetic separation.

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* Consulting Engineer.

† President, Separations Engineering Corporation.

Development of Magnetic Separation

Although the ancient Greeks were familiar with some of the phenomena of magnetism as early as 550 B.C., and though the Chinese used the compass according to authenticated references as early as 121 A.D., it was not until 1845 that Michael Faraday discovered that all substances were susceptible in a varying degree to the influence of a magnetic field. Edison, Wetherill and others applied this to the separation of iron ore from a gangue and carried the process further in the successful separation of some strongly magnetic minerals; they differentiated between "feebly magnetic" and "nonmagnetic" materials. The feebly magnetic were described as substances that an ordinary magnet would not affect perceptibly but which could be influenced by a very strong magnetic field, while the great majority of minerals were classed as nonmagnetic.

Up to 1925 Faraday's discovery that no substances were indifferent to the magnetic field had not brought about the extension of magnetic methods of separation to materials of which the susceptibilities were so small that they were classed popularly as nonmagnetic. To appreciate clearly how this old division between feebly magnetic and nonmagnetic substances has been wiped away, it should be understood that the force on a particle is proportional to the square of the field strength and that an ordinary permanent magnet has a field strength of the order of 800 gauss (lines of force per square centimeter) while modern magnetic separators employ field strengths as high as 15,000 gauss.

Magnetic separation, like nearly all other separating methods, depends on the density of the particles as well as on the particular physical property characteristic of the method. In these methods all the particles are acted on by two forces, one of which is gravity. The other force is in a direction different from gravity and therefore the *direction* of the resultant force is determined by the *ratio* between the special force employed and gravity. Although the special force may be and usually has been upward, directly opposed to gravity, it may be partly or entirely horizontal.

In magnetic separation both vertical and horizontal forces have been used. The older machines in general were vertical force types; the newer and more sensitive machines employ horizontal forces. The ratio of the forces is

$$\frac{\text{Magnetic Force per Unit Volume}}{\text{Gravitational Force per Unit Volume}} = \frac{ck}{gD}$$

where k = magnetic susceptibility of the particle,
g = gravitational constant = 980 dynes per gram,
D = density,
c = a constant depending on the geometrical and magnetic configuration of the machine.

$\frac{k}{D}$ is called the mass susceptibility and is the single physical constant characteristic of the substance comprising the particle which determines its behavior in a magnetic separator. In other words, the pull on a particle is affected by the degree of concentration of the magnetic field and by its intensity; it also is proportional to the magnetic susceptibility of the particle; and the usefulness of this pull must be gaged by its relation to the pull of gravity. With non-ferrous substances the susceptibility is usually a very small quantity, of the order 10^{-6} (for example, quartz) and even is negative in a few rare cases; *e. g.*, diamagnetic substances, such as bismuth. These figures are to be compared with a susceptibility of the order of 100 for iron. Thus to obtain separations of nonmetallics, one from the other, it is in general necessary to employ very intense and highly concentrated fields, and to use the forces exerted by these fields in the most efficient way to get a physical separation into two grain streams. The latter requirement may be met by using the magnetic forces to deflect particles in a falling stream, where a force equal to only a small fraction of the weight of the particle is able to produce a useful deflection, rather than by attempting to lift the particles bodily against gravity.

About 1926, Fred R. Johnson, working in the research laboratories of The Exolon Co., devised a separator combining high magnetic field intensity with deflection of a falling grain stream (Fig. 1). His machine showed remarkable results. It was able to make separations between two components, both of which were commonly considered nonmagnetic. His separation of biotite and muscovite mica from feldspar aroused great interest, which led to very active development work on the part of the company just mentioned and stimulated all the other builders of separators to the design of new and more sensitive machines. Muscovite mica and feldspar are two white materials which were both formerly considered nonmagnetic.

The U. S. Bureau of Mines refers to these cases in *Information Circular* 6488 (1931), as follows:

> The Johnson separators are of the electromagnetic induction type. Each machine has two large coils, an upper coil with three pole pieces at each end, and a lower coil with two pole pieces at each end. A laminated rotor, 30 inches long, revolves under each of these pole pieces. A special device feeds a thin uniform stream of the sized spar over the length of each rotor. As the material passes from one rotor to the next, all magnetic particles are deflected by the rotors, permitting the feldspar to continue on through five successive cleaning treatments. Each rotor removes a portion of the contaminating minerals, the more highly magnetic being removed by the upper rotors and the feebly magnetic by the lower rotors.
>
> The strength of the magnetic field developed by the Johnson induction separator is phenomenal, and as the separation depends on the deflection of magnetic particles in falling rather than on lifting, astonishing results are obtained, in that minerals even

with traces of iron are removed. For example, muscovite without visible iron stain is removed from the spar which, prepared by this method, consistently contains less than 0.06 per cent of iron or about one-half of the usual maximum specification for this type of spar.

METHOD OF OPERATION

At present it seems necessary to keep the size of feed 8 mesh or below. If the screen analysis shows the grains to be very close together in size, such as sand, it is not necessary to screen more closely than by taking all the material that passes through the scalping mesh. However, in most instances it has been found that a better product is recovered if the −8-mesh material is sized to various divisions such as −8 +16, −16 +30, −30 +60, −60 +100, etc. In general it is more difficult to handle extremely fine sizes, nevertheless in one instance an air separation product of −300 +400 mesh is being treated.

In the new machines a thin stream of properly dried and sized ore is fed down a chute and over a rotor. The rotor is highly magnetized by induction by a pole piece which faces the rotor across its entire length; the surface of the rotor is laminated, having alternate areas of iron and nonmagnetic material, so as to cause a concentration of the magnetic lines of force upon the iron surfaces of the rotor. It is this concentration which enables the magnetic field to act upon magnetic particles in the grain stream and to pull them towards the rotor surface, and good design in this respect is necessary to give the highest possible value to the constant c in the formula given above. The more highly magnetic particles in the stream are more deflected than the less susceptible ones and fall short of them, making two separate streams physically divided by a knife-edge splitting chute, which should have a calibrated setting. The stream that was less deflected is passed on over a series of rotors, each of which is progressively more highly magnetized than the last. The grain stream is split below each rotor, thus losing the particles that are sufficiently susceptible to be deflected by the stronger magnetic field encountered. Thus there is a magnetic fractionation of the original mixture and an end product is reached which contains substances that cannot or need not be separated further by magnetic means.

A feature of magnetic separation which is not found in most other methods is its ability to handle a wide variation of quality in the head feed without shifting splitter adjustments to any great degree. On a table, for instance, if an ore varies between 10 and 50 per cent rich, suitable adjustments would have to be made for this variation, while in this process the knife edge would not have to be adjusted and only a small difference in the purity of the concentrate would result.

The capacity of a machine in tons per hour is proportional to the width, thickness, velocity and density of the grain stream. The density, of course, is a property of the given material; the permissible velocity

and thickness of the grain stream depend on the design of the machine and on the material, and in general are decreased with increasing difficulty of separation. That is, if the particles to be separated are very feebly magnetic, or if their mass susceptibilities are very close together, either the velocity or thickness of the grain stream, or both, may have to be reduced. Therefore, as might be expected, the capacity of a machine decreases sharply with the difficulty of the separation.

In a well designed modern machine for nonmetallic separation the magnetic field is intense, the value of c in the formula is high and therefore the pull on the particle is high. The fact that a stream is merely deflected rather than lifted makes even a small force effective to separate. For these reasons the length of time that the particles must be in the zone of the magnetic force to acquire an effective separating momentum is much shorter than in the older types of belt machines, where the forces were vertical and were less strong in intensity, so that the tonnage capacities of the modern machine are very much greater than were attained formerly. In one instance, working on the same ore, a modern machine is handling a tonnage per foot width of grain stream almost four times as great as a machine built prior to 1925. The capacity of the standard size of one modern make varies on nonmetallics between 1 and 5 tons of feed per hour, depending on the factors stated above.

An example of interesting separation is the removal of ash from coal. Laboratory tests have lowered the ash content of anthracite coal from 26 to 11.3 per cent with a 70 per cent recovery. The ash content of bituminous coal has been lowered from 10 down to 6 per cent with 92 per cent recovery. As another example, a kyanite of 98.7 per cent purity is produced by magnetic separation followed by electrostatic separation. The magnetic separator recovers 68 per cent by weight of the feed, giving a recovery of 81 per cent of value, with a tailing content of 4 per cent kyanite. An interesting phenomenon is that in some instances a better separation is achieved by using a less powerful field than would be thought necessary, and in fact better than a far more powerful flux would yield. This may not always be due to the fact that both materials being separated are thrown one way by the more powerful magnet, but may sometimes be explained by the necessity of using the greatest possible difference in mass susceptibility between the two particles. This may be found in a less powerful field, since the susceptibility is not always constant for all field strengths.

Cost Data

The cost of separation varies from about $0.07 to about $0.60 per ton of head feed. This variation is due first to differences in fixed charges, since the cost of a machine depends in part on the strength of the magnetic field which it must produce to separate the particular materials.

If only low field strengths are required, as in separating ferromagnetic substances, the purely mechanical items preponderate in the cost of the machine, the copper and iron costs are low, and the power cost for energizing the magnets is small. However, for very feebly susceptible substances, large amounts of copper and iron are required in the electrical and magnetic circuits in order to produce strong fields. Machines have recently been built giving flux densities as high as 17,000 gauss. In such cases special iron must generally be used; also forced-draft cooling systems for the coils. One of the most recent advances has been

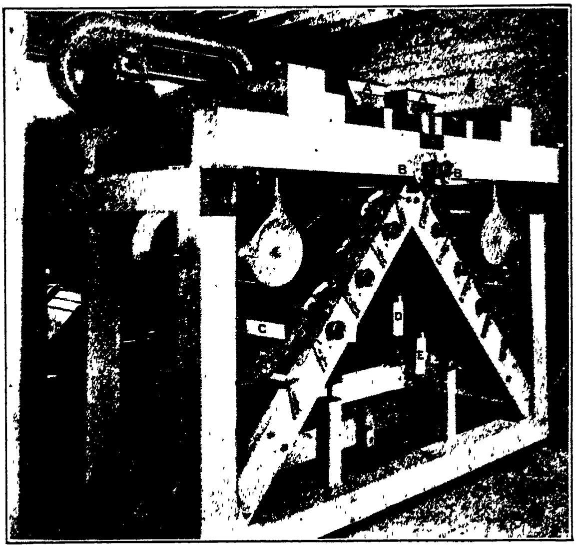

FIG. 1.—JOHNSON INDUCTION MAGNETIC SEPARATOR.
A. Feed hopper. *B*. Feed adjuster. *C*. Not removed, nonmagnetic. *D*. First removed, magnetic. *E*. Second removed, magnetic.

in the direction of economy of electric power consumed. The latest designs operate on from 1 to 5 kw-hr. per ton, depending on rate of feed and strength of field required.

Detailed data are not available at this time from all industries. However, it may be said that in one specific instance in a nonmetallic operation the following results are achieved per machine:

Tons per hour head feed	1.74
Tons per hour concentrates	1.62
Percentage recovery	93.86
Average percentage Fe_2O_3 in heads	0.12–0.15
Average percentage Fe_2O_3 in concentrates	0.045
Lowest Fe_2O_3 analysis in concentrates, per cent	0.036
Highest Fe_2O_3 analysis in concentrates, per cent	0.055

In the course of tests on different mineral mixtures, some peculiar effects have been found which cannot be clearly attributed to previously

known causes. For instance, an ilmenite running to all intents and purposes equal to another ilmenite in size, shape and chemical composition is much more difficult to separate magnetically. It has been observed repeatedly that the magnetic susceptibilities of samples of the same mineral vary when there is no apparent reason for such deviation, except geographical origin. An interesting surmise is that these differences in susceptibility may be due to different proportions of isotopes of the same element in two samples.

In addition to the actual separating costs, there is also to be considered the cost of preparation, which, again, varies with the nature of the material. In general all substances must be dried to the point where the particles do not adhere to each other. They must, of course, be ground down to the releasing point. Naturally, no material homogeneously mixed can be separated by a physical process. It must be a mechanical mixture, and that mixture must contain particles of such separate size that they can be released before grinding down into uneconomical minute sizes. For instance, if sugar is placed in dough there is a mechanical mixture, but the resultant mass must be ground down so fine that it cannot be separated economically. In difficult separations, where it is necessary to eliminate all possible interference with the very small magnetic forces developed on the particles, rather close sizing must be done to avoid the differential effects of air friction on particles of different sizes. These costs of grinding, drying and sizing usually will exceed the cost of separation.

Future of Magnetic Separation

In looking at the future of magnetic separation, we should bear in mind that it will be used as it is now being used, sometimes alone as a self-sufficient means and sometimes in conjunction with other methods to accomplish results impossible or uneconomical by any single means. It is reasonable to believe that research both in the design of machines and in their applications will continue in the next five years as rapidly as it has in the last five and make the magnetic method one of the most important tools in ore dressing.

The use of this new process is of particular interest to the ceramic industry for the removal of faintly iron-stained particles from their bodies. It is also of interest to all producers of nonmetallic materials such as bauxite, coal, fluorspar, kyanite, barite and the rare earths, etc. It would be well worth while for research engineers to look again into separating problems that have been given up as hopeless.

The magnetic separator is a dry concentrator; it is automatic, it is not dangerous, it is not affected by weather, its operation is simple, its cost per ton is low and its results are uniform.

DISCUSSION

(W. M. Weigel presiding)

F. A. JORDAN, Youngstown, Ohio.—Can you effect a separation on material of −200 mesh?

G. W. JARMAN, JR.—Separation is being made of some materials between 350 and 400 mesh.

F. A. JORDAN.—What is the limiting moisture content?

G. W. JARMAN, JR.—This will vary in different materials, as it depends entirely on whether the particles cling together or not. It should be low enough so that there is no clinging, and ordinarily is one-half of one per cent perhaps.

MEMBER.—Is that the amount of moisture contained in feldspar which is to be separated?

G. W. JARMAN, JR.—Yes.

MEMBER.—What is the recovery?

G. W. JARMAN, JR.—About 90 per cent.

F. W. LEE, Washington, D. C.—Did you make any determination of the susceptibility beyond which the machine cannot operate?

S. G. FRANTZ.—We believe we can pull out materials with a mass susceptibility as low as 10^{-6} C.G.S.

C. Q. PAYNE, New York, N. Y.—What is the diameter of the rotor, and at what speed does it revolve?

S. G. FRANTZ.—It is 4 in. in diameter and turns over at 60 to 150 revolutions per minute.

F. A. JORDAN.—What is the largest size of particle that can be treated?

S. G. FRANTZ.—The largest actually being treated now is 8 mesh.

F. A. JORDAN.—What is the capacity of the machine?

S. G. FRANTZ.—The capacity depends on the size and nature of the material being treated. A 30-in. rotor will handle 1000 to 5000 lb. per hour.

MEMBER.—Do you have to keep the material in one layer?

S. G. FRANTZ.—Not in the chute. I am not sure what happens on the roller.

MEMBER.—I can answer that question. If a large proportion of the material is magnetic, there must not be many layers, as the magnetic material would hold the nonmagnetic material on the roller with it.

F. A. JORDAN.—What is the speed of the feed?

S. G. FRANTZ.—Little difficulty is experienced in the matter of speed with which material is fed to the machine. It may be varied, the only thing is to avoid feeding so fast as to cause bouncing in the chutes.

C. Q. PAYNE (written discussion).—This paper is very welcome as indicating a possible revival of interest in the subject of magnetic separation. I regret, however, that the information supplied in regard to the Johnson separator is so limited that it is difficult to form any idea of its design and operation.

The principle difference between the Johnson separator and others of the high-intensity type, which accomplish similar results, seems to be that it has a somewhat larger capacity, and that it has been applied mainly to the treatment of material like feldspar, in which the nonmagnetic portion, usually the tailing, is here the concentrate.

Judging from the paper by Messrs. Coghill and Clemmer,[1] there appears to be an emerging conflict, or at least a twilight zone between magnetic separation and soap flotation in the beneficiation of certain minerals. Both papers mention the concentra-

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[1] W. H. Coghill and J. B. Clemmer: Soap Flotation of the Nonsulfides. A.I.M.E *Tech. Pub.* 445 (1932).

tion of kyanite by these different methods. Ultimately, each method is likely to develop its own special fields, and applications will also be found in which both can cooperate to the general advantage. Having done a good deal of work in magnetic separation in the past, my interest is enlisted on this side of the contest.

G. W. Jarman, Jr. (written discussion).—We would like to qualify Mr. Payne's interesting discussion in one regard; namely, that "it has been applied mainly to the treatment of material like feldspar." While this is about the general type of separation which the Johnson separator inaugurated, nevertheless it is incorrect to suppose that the Johnson is mainly confined to feldspar or to materials which have the concentrate as a nonmagnetic. There are installations in other fields in which the Johnson separator fractionates material, producing a concentrate which is susceptible to its magnetic attraction, and another concentrate which is not magnetic. For instance, operating on nelsonite it produces an ilmenite concentrate and an apatite concentrate and also a micaceous material which is of no value, but there are three distinct products.

Mining and Treatment of the Sillimanite Group of Minerals and Their Use in Ceramic Products

By Frank Harwood Riddle,* Detroit, Mich.

(New York Meeting, February, 1932)

Products made from the ores of the sillimanite group, and synthetic substitutes for them, have unique properties, and service tests prove that they are playing, and will continue to play, a major part in improvements in metallurgical processes. Also, they are examples of how comparatively useless, rare, or "museum" minerals have been made available in commercial quantities. Until research developed uses for them, the minerals had been observed only in small, scattered masses, or crystals. Improvements in ceramic processes and compositions have come about very gradually. In the last 10 or 15 years, several outstanding developments have been made. Of greatest interest to the mining and metallurgical engineers are those affecting the composition, necessitating the mining of new minerals, and improvements in ceramic products used in metallurgical processes.

Sillimanite and Mullite

Sillimanite ($Al_2O_3.SiO_2$), then known as Fibrolite, according to Dana,[1] was first reported in 1792. In 1796, cyanite ($Al_2O_3.SiO_2$) was first described.[2] In 1873, Behrens published results of his microscopic study[3] of porcelain. He showed that it contained what appeared to be crystals, which dissolved less readily than quartz and glass in hydrofluoric acid.

Hussak[4] described the presence of needlelike crystals as occurring in some instances in small amounts in certain porcelains. In 1908, Plenske and Zoellner both studied porcelain structure.[5] Plenske observed that "sillimanite" might be present in masses of exceedingly minute grains which apparently were amorphous. Zoellner advanced the theory that the needlelike crystals were formed by a molecular change in the clay,

* Director of Research, Champion Porcelain Co.

[1] Mayer's *Samml. phys. Aufs.* (1792) **2**, 277; *Bergm. Jern.*, (1792) **2**, 65.

[2] Saussure: Voyage dans les Alpes, **4**, 84–5. Neuch., 1796.

[3] H. Behrens: Ueber das porzellan und einige verwandte entglasserungsproduckte. *Pogg. Ann.* (1873) **150**, 386.

[4] E. Hussak: *Sprechsaal* (1889) 153.

[5] E. Plenske: Ueber mikrostruktur und bildung des porzellans. *Tonind. Ztg.* (1908) 1343; *Sprechsaal* (1908) 41 (Nos. 20, 21, 22).

A. Zoellner: Zur frage nach der chemische-physikalischen natur des porzellans. *Chemische Industrie* (1908) 212.

and noted that they had petrographic characteristics similar to those in the natural mineral sillimanite. Many others made observations along similar lines.

About 1917 ceramists at the United States Bureau of Standards were given the problem of developing a better spark plug core. They decided to experiment with sillimanite, but because of the scarcity of the natural minerals of this group, they decided to prepare synthetic sillimanite. It was made by intimately grinding clay ($Al_2O_3.2SiO_2.2H_2O$) and Al_2O_3 in the proper proportions to form sillimanite ($Al_2O_3.SiO_2$). The combination was fired to a temperature [cone 20, approximately 1550° C. (2822° F.)] high enough not only to produce sillimanite from the clay but also to unite the added alumina with the silica remaining from the dissociation of the clay to form additional sillimanite.

This sillimanite calcine was deliberately incorporated into a porcelain body in a quantity larger than it would have been possible to develop from the clay contained in the body. It was used as a substitute for other ingredients, especially quartz or potter's flint. The latter, because of its very large expansion and conversion from the alpha to the beta form at elevated temperatures, and its reversible transformation during cooling, brought about excessive volume changes.

The addition of the sillimanite calcine to the body to replace potter's flint, and the firing of the body to a temperature high enough to develop sillimanite in the clay, resulted in a body with a mechanical strength two to four times as great as normal porcelain. It also improved heat-shock resistance and made a much better spark plug core, particularly when alkali earths were substituted for alkali as a flux, a change which greatly increased the dielectric value of the product when hot.[6]

A study of the natural mineral showed its melting point to be 1810° C. (3290° F.). Other studies showed sillimanite to be constant in volume and to possess characteristics that would be beneficial if it were used as a refractory ingredient. A suitable refractory was made, tested and shown to possess favorable possibilities.[7] The successful use of synthetic sillimanite caused manufacturers to search for the natural mineral in commercial quantities. Obviously a suitable natural mineral would be less expensive and more stable than the artificial substitute.

Among the minerals having the chemical composition $Al_2O_3.SiO_2$ are andalusite, cyanite and several forms of sillimanite, including chiastolite, fibrolite, bamlite, zenolite and worthite. All of these are spoken of as minerals of the sillimanite group. Dumortierite ($8Al_2O_3.6SiO_2$.-

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[6] Properties and Preparation of Ceramic Insulating Materials for Spark Plugs. National Advisory Committee for Aeronautics, Report No. 33 (1920); also Bleininger and Riddle: Special Spark Plug Porcelains. *Jnl.* Amer. Ceram. Soc. (1919) **2**, 564.

[7] A. V. Bleininger: Note on the Load Behavior of Aluminous Refractories. *Jnl.* Amer. Ceram. Soc. (1920) **3**, 155–57.

TABLE 1.—*Important Optical and Physical Properties of Minerals of the Sillimanite Group*

All values except those for mullite are taken from Larsen: U. S. Geol. Survey *Bull.* 679. Those for mullite are taken from Bowen and Greig: The System $Al_2O_3.SiO_2$, *Jnl.* Amer. Ceram. Soc. [4] (1924) **7**, 238.

Theoretical Composition of Mineral	Indices of Refraction			Optical Character	Crystal System and Habit	Cleavage	Hardness and Specific Gravity	True Optic Angle
	α	γ	β					
Mullite $3Al_2O_3.2SiO_2$	1.642	1.654	.....	Positive	Orthorhombic prisms and needles	(010) perfect	Sp. Gr. = 3.156	45° to 50°
Sillimanite $Al_2O_3.SiO_2$	1.659	1.680	1.660	Positive	Orthorhombic prisms and needles	(010) perfect	H = 7, Sp. Gr. = 3.23	20° ±
Andalusite $Al_2O_3.SiO_2$	1.632	1.643	1.638	Negative	Orthorhombic elongated crystals	(110) perfect	H = 7.5, Sp. Gr. = 3.2	83°–85°
Cyanite $Al_2O_3.SiO_2$	1.712	1.728	1.720	Negative	Triclinic blades	(100) very perfect (010) less perfect (001) parting	H = 4 parallel to lgth. H = 7 parallel to wdth. Sp. Gr. = 3.6	82°
Dumortierite $8Al_2O_3.B_2O_3$ $6SiO_2.H_2O$	1.678	1.689	1.686	Negative	Orthorhombic needles	(100) distinct	H = 7, Sp. Gr. = 3.3	30°–40°
Topaz $Al(O,F_2)AlSiO_4$	1.619	1.627	1.620	Positive	Orthorhombic elongated crystals	(001) perfect	H = 8, Sp. Gr. = 3.58	Varying

$B_2O_3.H_2O$), although containing more alumina and some fluxes, is similar in many ways. Other minerals that might have possibilities are topaz, $Al(O, F_2)AlSiO_4$, and zunyite, $(Al(OH, F, Cl)_2)_6Al_2Si_3O_{12}$.

Minerals of the sillimanite group, while of the same empirical chemical formula, differ crystallographically. Two of them, andalusite and sillimanite, are orthorhombic and form somewhat similar crystals which, however, can be differentiated by the accurate measurement of their interfacial angles. The third, cyanite, is triclinic, having a typical crystal form and certain distinctive physical properties. The optical constants of these minerals, however, allow complete differentiation to be

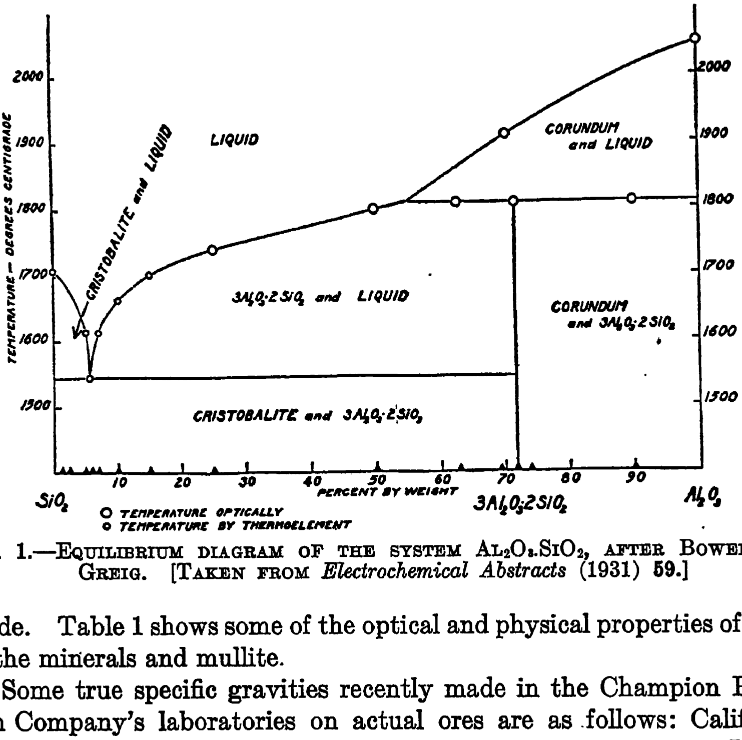

FIG. 1.—EQUILIBRIUM DIAGRAM OF THE SYSTEM $Al_2O_3.SiO_2$, AFTER BOWEN AND GREIG. [TAKEN FROM *Electrochemical Abstracts* (1931) 59.]

made. Table 1 shows some of the optical and physical properties of some of the minerals and mullite.

Some true specific gravities recently made in the Champion Porcelain Company's laboratories on actual ores are as follows: California andalusite, 3.025 and 3.085; South Dakota andalusite, 3.109; Indian sillimanite, 3.174; South Dakota sillimanite, 2.99; Nevada dumortierite, 3.239; Indian cyanite, 3.646.

In 1923–1924, Bowen and Greig showed that in artificial melts[8] containing Al_2O_3 and SiO_2 in varying proportions, the compound Al_2O_3.-SiO_2 corresponding to natural sillimanite is not formed, as had been stated in earlier work. The crystalline compound formed has the composition $3Al_2O_3.2SiO_2$. They have named this compound mullite.

[8] N. L. Bowen and J. W. Greig: The System $Al_2O_3.SiO_2$. *Jnl.* Amer. Ceram. Soc. (1924) **7**, 238–54.

Its physical and optical properties are similar to those of sillimanite; hence the consistent error by investigators. This similarity is shown in Table 1.

A study of the equilibrium diagram of the system $Al_2O_3.SiO_2$ (Fig. 1) shows that, regardless of the proportions of silica and alumina used, the final product, after exposure to sufficient heat, is synthetic sillimanite or mullite, and an excess of the predominating oxide determines the composition of the remaining portion. If the excess is silica, it will form a

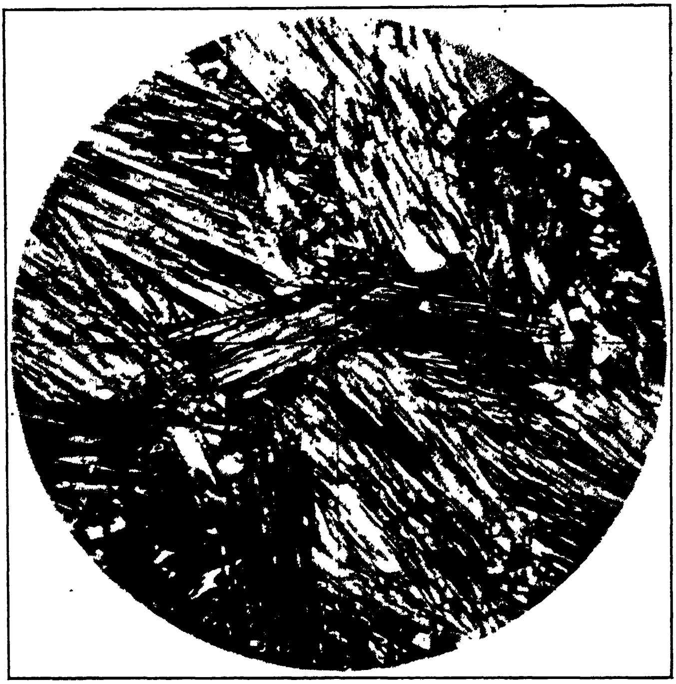

FIG. 2.—THIN SECTION OF SYNTHETIC SILLIMANITE OR MULLITE FORMED IN CERAMIC BODY; SUBSTANTIALLY 100 PER CENT CRYSTALLINE. (PHOTOMICROGRAPH BY T. S. CURTIS.)

glassy matrix which, if in too great an excess, will permit movement of the mullite crystals and deformation of the product. Obviously clay or kaolin ($Al_2O_3.2SiO_2.2H_2O$) when converted to mullite ($3Al_2O_3.2SiO_2$) has too much excess silica glass, as shown by the formula $3(Al_2O_3.2SiO_2.2H_2O)$, and heated above the conversion temperature produces $3Al_2O_3.2SiO_2$ and 4 excess SiO_2 glass. This excess glass is 36 per cent of the total weight. The sillimanite minerals ($Al_2O_3.SiO_2$) also have some excess silica glass, but only slightly over 12 per cent, which is just enough to produce a good glassy matrix and not enough to cause deformation at high temperatures. If sufficient corundum is added to a mix containing silli-

manite minerals, the corundum and the 12 per cent excess silica will combine to form additional mullite, and this crystallization will continue to improve as long as the silica continues to be available. Can you think of a better combination for a refractory? The longer the product is exposed to the action of heat, the greater the mullite formation and the better the product. This is an outstanding property of these products.

Natural Minerals

Andalusite

Dr. Joseph A. Jeffery, president of Champion Sillimanite, Inc., in his search for natural minerals followed a lead mentioned by Knopf[9] and located a deposit of andalusite two miles distant from Knopf's location. This deposit has been described by Peck[10] and others, and is of interest for several reasons. It is located at an elevation of about 10,000 ft. on the southwestern slope of White Mountain, in the Inyo Range, Mono County, California. Its inaccessibility makes necessary the carrying of the ore by packmules down the trail four and one-half miles, to a point from which a truck is used to transport the ore three miles to the narrow-gage railroad. Here it is placed in stock piles of 2000 tons each. The piling is done in parallel, adjoining rows. When it is withdrawn, the ore is removed in layers at right angles to the original piling, which assures a large tonnage of ore of uniform chemical composition. The truck is easily loaded through a chute from the bin in which the ore is placed after it has been brought down the hill on the packmules.

Fig. 3 shows the mine workings. These are reached by a trail cut in the perpendicular face of the cliff. The ore is mined from large chambers and stopes, which are connected by a series of tunnels. No timbering is necessary, since the orebody is compact and free from faults or slips. The ore is broken down by drilling and shooting with 40 per cent nitroglycerine, then cobbed and sacked for packing down the trail.

The apparent specific gravity of the ore varies directly with the silica and alumina content. For this reason, the ore is mined and blended according to Jolly balance tests run at the mine at frequent intervals. The grade of the ore in various workings determines the daily percentage of ore mined from each part in order to maintain a uniform mixture of ores at the stock pile.

On the whole, the rock material is a rather pure andalusite with comparatively small amounts of several other minerals. Samples are taken over the face of the working and, if satisfactory, the material is broken out, carefully hand selected, and sacked. Generally it runs at least 75 to 85 per cent andalusite upon microscopic examination. A

[9] A. Knopf: *Jnl.* Wash. Acad. Sci. (1917) **7**, 549.

[10] A. B. Peck: *The American Mineralogist* (June 1924) **9**.

typical chemical analysis from a sample representing a carload shipment recalculated to an Al_2SiO_5 basis will show over 90 per cent Al_2SiO_5. For example:

	Per Cent		Per Cent
SiO_2	33.78	Equivalent to	
Al_2O_3	56.89	Al_2SiO_5	94.18
H_2O	0.37	SiO_2	0.22
Ign. loss	3.67	Others	5.60
Other determined constituents	5.37		
	100.08		

The fact that the chemical analysis generally shows higher Al_2SiO_5 than is indicated by the microscopic examination is due to the very

Fig. 3.—Entrance to andalusite workings, with mine buildings in foreground. This property owned and operated by Champion Sillimanite, Inc. Photograph by Dr. Paul F. Kerr.

general presence of corundum in the material. In no case do the associated minerals assume any large proportion over any considerable area, although locally they may occasionally outweigh the andalusite.

Minerals Associated with Andalusite

The associates of the andalusite, as described by Peck,[11] are also deserving of mention, especially because several are counted among the less common minerals.

Lazulite is perhaps the most unusual mineral associated with the andalusite. It occurs in small, blue-green veinlets or masses scattered through the mass. Opti-

[11] A. B. Peck: Notes on Andalusite from California. *Rept. XX* of State Mineralogist, California State Mining Bureau (1924) **20**, No. 2, 194.

cally it shows a distinct light blue pleochroism. Chemical analysis usually shows small amounts of P_2O_5 present in the rock.

Pyrophyllite is another uncommon mineral found with the andalusite. It occurs in crusts of radial fibers on the walls of cracks, or as radial masses in small cavities. Owing to its chemical composition, as soon as its water is lost it takes on nearly the same composition as andalusite, and hence can hardly be considered as an impurity.

Muscovite also occurs in much the same manner as pyrophyllite, and is somewhat more abundant, but at no time reaches large proportions. It occurs in distinct plates, usually in divergent groups.

Corundum is a rather common associate of the granular andalusite, and is usually deep sapphire blue in color. It occurs generally in small scattered plates or grains, but occasionally in lenses two or three inches thick. According to observation thus far, it is always blue in color. A few crystals of andalusite with blue corundum centers have been noted. For manufacturing purposes, a small amount of corundum is favorable to the neutralization of any excess quartz in the rock.

Rutile is a rather constant associate, largely as microscopic inclusions in the andalusite. At times, small, free crystals are found. This constant association of rutile and the blue color of the corundum seem to confirm the theory that the color of the sapphire is due to the presence of TiO_2.

Pyrite is sometimes found locally, especially near open veins. When near the surface, it has often weathered, leaving a stained and porous rock.

Barite also has been noted a few times, one large, right-angled twin having been found.

Lazurite also has been found, not in the andalusite but in veins of milky quartz adjacent to the mass. It is interesting to note that Knopf points out that the occurrence examined by him across the canyon and to the southwest was first staked as a silver mine, the bright blue lazurite being mistaken for AgBr. After an assay showed no silver, the claim was dropped. Later the brown andalusite mass was staked again, this time under the impression that it was apatite. This, in turn, proved valueless, and this claim was relinquished.

None of these associates are objectionable in porcelain cores for spark plugs; in fact, the fluxes present are beneficial, particularly as no soda or potash are present in appreciable amounts. For this reason the mining of the ore for porcelain has permitted the use of ore of lower grade, while for refractories the fluxes are more objectionable, and the mining is done accordingly.

Two thousand Jolly balance samples are run each operating month. The samples used for these tests are saved, averaged, and tested each month in the laboratories of the Detroit plant. This is one of many plant control tests, the use of which has resulted in the elimination of losses and the development of many technical sillimanite products.[12]

Dr. Jeffery discovered the deposit in 1919, and opened it in June, 1922. Since then the Champion Porcelain Co. has manufactured approximately 350,000,000 spark plug cores in which andalusite has been by far the major constituent. These cores are fired to temperatures (1450° C., 2642° F.) higher than those usually employed in burning ceramic products (1225° C. to 1285° C., 2237° to 2347° F.), and most of the refractories,

[12] F. H. Riddle and H. F. Royal: Process Control in Continuous Production. *Ind. & Eng. Chem.* (1930) **22**, 14.

particularly those exposed to the highest temperatures in the manufacture of these cores, are made from andalusite.

Without the andalusite for refractories, several processes that have made marked improvements in the spark plug cores would not have been possible. These results encouraged experimental work with the refractories in other fields. This will be discussed later.

Although there are outcrops of andalusite in several localities in the United States and other countries, this is the only deposit where andalusite occurs in sufficient quantity to be of commercial value. Conservative estimates show several million tons of ore available. It is possible that other deposits may be discovered.

Treatment of Andalusite

When the ore is received at the plant in Detroit, it is treated by one of two methods, depending on the use to which it is to be put. For refractories, the ore is crushed to 1 in. and smaller in a jaw crusher, and ground in a Herman mill. The Herman mill was selected to eliminate excess fines as much as possible. The desirable grain sizes for many types of refractories are about equal proportions by weight of 8 to 14 mesh, 14 to 30 mesh and 30 mesh and finer.

The crushed ore, after passing over a magnetic separator, is sized through a Tyler Hum-Mer screen and then distributed to storage bins. When used it is proportioned and mixed with clays or other aluminous binders, formed into desired shapes by several methods and fired to at least cone 16 (1450° C., 2642° F.) or higher. When plastic patches or ramming mixes are made, the grain, some of which is fairly coarse, the binder, and a definite amount of water are thoroughly blended in suitable mixers. These mixes are shipped in sealed containers, because the success of their application depends partly on their physical condition at the time they are used. Ramming mixes are proving to be of considerable importance in metallurgical processes.

When andalusite is used in porcelain and bodies of similar type, fine grinding is essential. The crushed ore is reduced to 8 mesh and finer in a Hardinge ball mill, passed over a magnetic separator and then ground in a Hardinge pebble mill. The latter is lined with sillimanite porcelain and the grinding is done with sillimanite balls. The final product is 30 mesh and finer, 60 per cent being finer than 325 mesh.

For body making, the ground andalusite, compounded with the other body ingredients such as the flux and clays, is mixed in pebble mills with sufficient water to form a slip or creamy suspension, and ground for 19½ hr. It is then fine enough to pass through a 325-mesh lawn. The slip is passed through a lawn, over a magnetic separator, and then filter-pressed. Enough water is eliminated so that the plastic body can

be handled and stored in an aging cellar for use in forming ceramic products. This process has been described in detail elsewhere.[13]

The andalusite ore is of particular value because it has no appreciable volume change when converted to mullite. This makes it possible to use it raw; *i. e.*, without previous calcination. This is not merely a financial saving. When calcined, the grains tend to crystallize in themselves and become satisfied (chemically). Raw grains in a body, when calcined, tend to grow together; that is, the crystals seem to knit so that the final product has an excellent bonded strength, making it particularly valuable for refractories designed to be used under sag loads at increased temperatures.[14]

to a temperature of 1700° to 1800° F. This reduces the iron oxide for later removal by magnetic separation and converts alpha to beta quartz. The calcine falls from the kiln into water for quenching and rapid chilling, which converts the quartz back again to the alpha form. This quartz conversion causes a sudden expansion and a like contraction, while the cyanite shows only a slight expansion and contraction. The quartz is shattered loose. The ore is then crushed in a swing-hammer crusher and washed over slightly inclined shaking screens. The relatively pure cyanite fibers will remain on the screen and the quartz gangue will pass

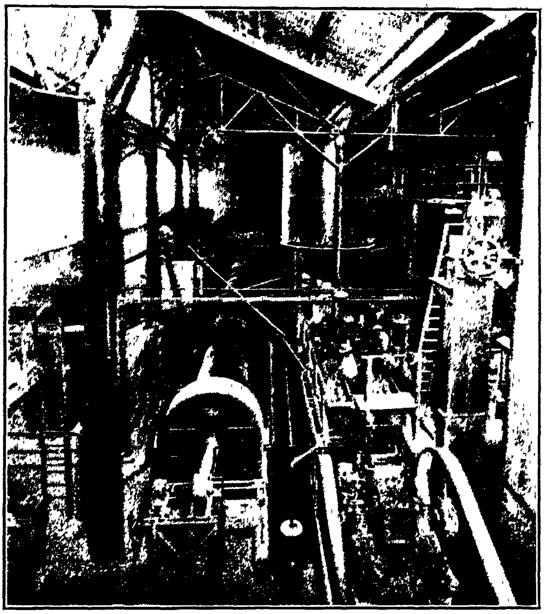

FIG. 4.—ORE-TREATING EQUIPMENT AT VITREFRAX PLANT, LOS ANGELES, SHOWING ROTARY KILN AND SORTING SCREENS.

through as sand. This separation method is also used by Vitrefrax Corporation for refining pyrophyllite and some other aluminum silicates.

The refined cyanite is used in several ways, depending on the product desired.

A more massive cyanite is being imported into the United States by the Chas. F. Taylor and Sons Co., of Cincinnati, Ohio, which is known under the trade name "P. B. Sillimanite," after the owners, Paule and Bredick. This occurrence is in the State of Kharsawan, Province of Bihar and Orissa, about 100 miles west of Calcutta, and consists of large, well-rounded boulders which have been transported from their original, undiscovered source. The rock is chiefly cyanite, but there is some sillimanite and occasionally andalusite. This ore does not expand so much

when calcined as the cyanite from the southern part of the United States, although it does expand much more than andalusite, dumortierite, or sillimanite. It is stated that there are several million tons of this ore available, and it is being shipped to Germany and England, as well as to the United States. The boulders are hauled in ox-carts 10 miles to the nearest railroad.

The ore is shipped to Cincinnati, Ohio, in the original boulder form. There the boulders are piled into a round down-draft brick kiln, and calcined to a temperature well above the conversion temperature of cone 12. Calcining the ore requires several days and not only produces a product that will be free from any conversion volume changes in later burns but also breaks up the boulders so that they are easily crushed and ground to desired sizes for manufacturing processes.

Sillimanite

Sillimanite appears to be much more scarce than any of the other minerals of the group. Several small outcroppings have been opened in the United States, chiefly in South Dakota. Several weeks' work recently resulted in the gathering of only two tons of such ore.

Sillimanite has been discovered in India, where it is said to occur in rather large amounts. The deposit is situated within the boundaries of the village of Mongmaweit in the Khasia State of Nongstion in the N. W. Khasia Hill. It is 52 miles from Palasbaria, the steamer station on the River Brahmaputra, and 21 miles south of Heima at the foot of the hill. There is a 31-mile motor road from Palasbaria to Heima. From Heima to the mines, there is a 21-mile mule track rising 3400 ft. over the hills. It would seem that ore from this deposit will not be on the market for some time.

Dumortierite ($8Al_2O_3.6SiO_2.B_2O_3.H_2O$)

Dumortierite seems to have first been found by Ferdinand Gonnard in 1879, near Lyon, France.[16]

In 1880, Bertrand published the first article about dumortierite.[17] He realized that it was an unknown mineral—not cyanite, as at first supposed by Gonnard.

In 1881, Gonnard named the mineral dumortierite, after Eugene Dumortier, a famous paleontologist of Lyons. Minor deposits have been found in many parts of the world.

A deposit of dumortierite was opened in 1925. It is owned and operated by Champion Sillimanite, Inc., and used exclusively by the

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[16] F. Gonnard: Note sur l'existence d'une espèce minérale nouvelle, la dumortiérite, dans le gneiss de Beauman au-dessus des anciens aqueducs galloromains de la vallée l'Iseron, Rhône (*Bull.* Soc. Min. de France (1881) **4**, 2–5.

[17] E. Bertrand: Sur un minéral bleu de Chaponost, prés Lyon. *Bull.* Soc. Min. de France (1880) **3**, 171–172.

Champion Porcelain Co. It is located in the Humboldt range, in Humboldt Queen Canyon, near Oreana, Nev., which is on the main line of the Southern Pacific Railroad.

In 1926, petrographic examination of this ore was made by Peck.[18] The ore contains some impurities, such as small amounts of quartz and paragonite and larger amounts of muscovite mica. In 1928, the Staff of the Mackay School of Mines published an interesting bulletin on dumortierite.[19]

The Oreana Deposit

The geology of the Oreana deposit, and the development work on it, are clearly described by Dr. J. C. Jones, Professor of Geology and Mineralogy at the Mackay School of Mines. Parts of his description follow:

> Summarizing the results of the present study, it may be stated that the dumortierite used from this locality is won from lenticular masses occurring in zones formed by solutions originating in a granitic magma that intruded the region. While dumortierite also occurs in quartz veins, as in other known localities, yet the lenticular masses originated through the replacement of pre-existing lenses of andalusite formed during the earlier stage in the metamorphism of the rocks by the granitic magma. During the period of replacement of the andalusite little quartz was deposited, the quartz-dumortierite veins forming later. Thus the lenses contain only the tolerated minerals and can be utilized, while the veins are worthless.
>
> The deposits of dumortierite in Humboldt Queen Canyon occur near the head of the canyon at an elevation of approximately 5500 ft. above sea level and about 1000 ft. above Oreana. The road from Oreana leads across the flat-floored Humboldt Valley to the foot of the mountains, then rises rapidly due east through the steep-walled canyon. At the property of the Champion Porcelain Co., the canyon branches, the main canyon turning south, with a branch to the north and a minor saddle continuing east.
>
> The dumortierite occurs in two parallel zones striking east of north and dipping west. The western zone, at the fork of the canyon, is marked by a bold outcrop of massive quartz several hundred feet long, 50 ft. wide, striking N. 20° E., and dipping 55° westerly. Little development work has been done upon it as yet. The eastern zone is about 600 ft. east from the western zone and continues up the northern branch of the canyon. This zone is 75 ft. wide, strikes N. 15° E., dips 50° W., and has furnished the bulk of the dumortierite obtained up to the present time. North and south of the saddle, the zone is marked by bold outcrops of massive quartz through which are numerous stringers and veins of quartz and dumortierite. At the saddle, however, the zone is inconspicuous and large boulders of nearly pure, massive dumortierite were found along and below the zone. The greater part of the production has come from these boulders and lenses found in and near the saddle. It is evident that the saddle has been formed due to the somewhat softer character of the rocks in this portion of the zone.

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[18] A. B. Peck: Dumortierite as a Commercial Mineral. *Amer. Mineralogist* (1926) 11, 96, 101.

[19] A Bulletin by the Mackay School of Mines Staff on the Mineral Dumortierite. Univ. of Nevada *Bull.* (1928) **22.**

The deposits have been explored by numerous open cuts and shallow tunnels in the zone as it crosses the saddle and by about 1000 ft. of work in the lower tunnel driven to intersect the zone at depth. The surface workings (Fig. 5) follow the zone up the gulch to the north and it is apparent that the masses of high-grade dumortierite are most abundant in an elongate area or shoot pitching about 35° to the south. This shoot lies on the southern end or toe of the large outcrop of massive quartz to the north.

The lower tunnel cut the dumortierite zone about 160 ft. from the portal and at 180 ft. intersected a diabase dike that at the time seemed to be the footwall of the zone (Fig. 6). The tunnel then was turned to the north and followed the dike for 300 ft. At this point it became apparent that the dike had crossed the zone at a low angle and a crosscut was driven east through the zone behind the dike. This crosscut is under the southern end of the massive quartz outcropping up the northern

Fig. 5.—Surface workings at dumortierite deposit.
Arrows point to a lense of dumortierite in place. Ore at right of arrows has been removed.

branch of the canyon. For the first few feet the crosscut is in altered schist that forms the footwall of the zone. While the massive quartz contains numerous stringers and small lenses of dumortierite, muscovite, and quartz, none of the large masses of pure dumortierite were encountered.

A second crosscut was then driven where the tunnel first encountered the dike and on breaking through a large lense of dumortierite was found. This lense was spindleshaped and about 38 ft. long, 15 ft. high and 7 ft. wide, tapering at both ends. The longer axis pitched 35° to the north, lying across the general pitch of the shoot as disclosed in the surface openings. The lense lay on massive quartz with schist hanging walls and stringers of dumortierite leading from the lense. This seems to be the common mode of occurrence of the lenses found in place near the surface in the saddle.

Since Dr. Jones's article was written, mining operations have continued. Ore has been taken from both the surface workings and the lower tunnel where it cut the dumortierite zone. Over 2500 tons of ore have already been mined and shipped.

This ore is shipped to Detroit and treated in the same manner as the finely ground andalusite. It burns to a pure white, and the fluxes,

particularly the boron, have a beneficial effect upon the properties of the spark plug cores and chemical laboratory porcelain. This addition of dumortierite also improves or widens the burning range of the bodies. It is not at present used in the manufacture of any other products,

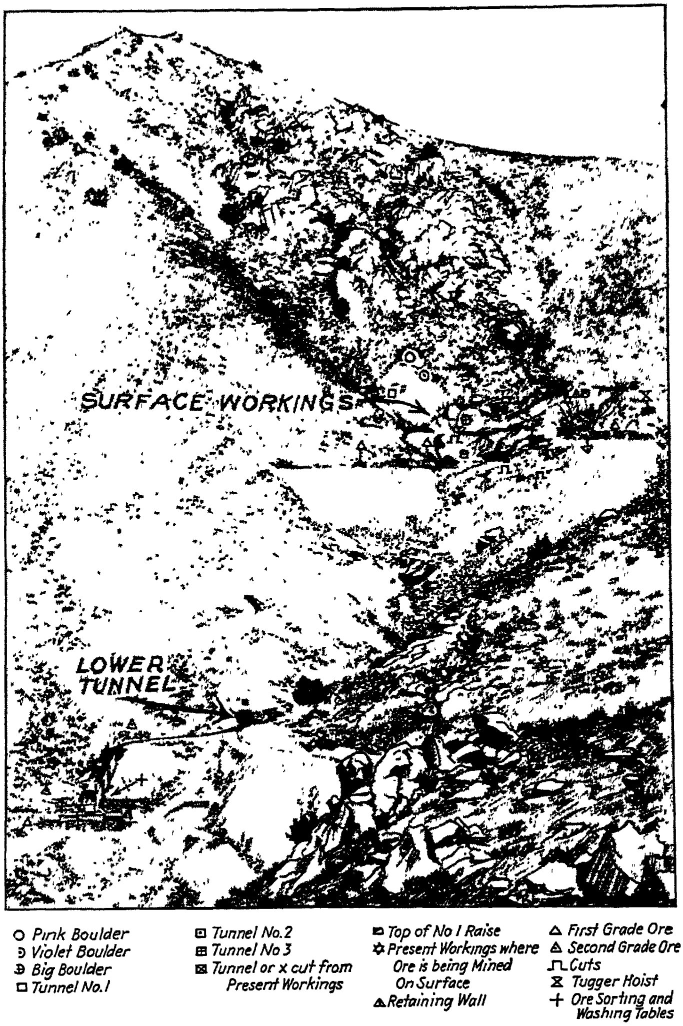

FIG. 6.—DUMORTIERITE SURFACE WORKINGS AND TUNNELS.
Reproduced from a pencil sketch by J. M. Cole, Superintendent.

although many experiments show it to be valuable in several types of refractories and in other ceramic bodies. Dumortierite converts to mullite and glass at a much lower temperature than any of the sillimanite minerals.

Synthetic Mullite

Synthetic mullites have been developed to a point where they are of equal importance with the natural minerals for use as refractory products. The Vitrefrax Corporation of Los Angeles, under the direction of Thomas S. Curtis, has made many interesting developments. It has produced mullite by the complete melting, in an electric-arc furnace, of various alumina-silica mixtures, forming "pigs" which are crushed to grain and fabricated in the usual way. The company also has a Curtis converter process which is much more economical to operate and which produces a more granular mullite than the cast brittle product. The charge, which may be finely ground cyanite and alunite or other alumina-silica mixtures suitably proportioned to form mullite, is made into briquets, dried, and placed in the converter or brick-lined cylinder which is balanced on trunnions at the sides and open at the top end. An opening is made at the other end and so connected that gas, air, or a mixture of them can be introduced into the furnace. Gas is introduced and lighted at the top of the furnace. Air is then added in sufficient volume to produce an explosive mixture. This causes a mild explosion, which extinguishes the free flame at the top and promotes combustion inside the converter, causing a roaring sound. After the first 2 or 3 hr., some flame is again visible on the surface, and after 4 hr. a settling of the charge is evident. As the charge settles, additional charge is added. After 12 or 14 hr., a cover is placed on the converter to develop all the heat possible at the top in order to treat properly the charge in this location. This condition is continued for a total of 19 hr. from the start, and is spoken of as the ignition stage.

The second, or air-blast, stage is then started. The gas is turned off, air alone being admitted for 2 hr. or so. This produces a distinct rise in temperature to cone 40 (3500° F.) causing noticeable settling of the charge and an additional conversion of the material that experience has shown is necessary. The converter is tipped on the trunnions, and the charge dumped into a chilling pit; this requires an hour or so. The final product is sized and mixed with suitable clays in a manner similar to that used in handling natural ores. The control of the ingredients of the charge and the heat treatment make a flexible economical process.

Refractory products manufactured by this company are marketed under the trade names "Argon" and "Durox." It also markets other products, including the partly converted cyanite described in a preceding paragraph.

The Corhart Refractories Company

A unique process is that of the Corhart Refractories Co., in Louisville, Ky. Approximately three years ago, this company, a subsidiary of the

Corning Glass Works and of the Hartford-Empire Co., started the successful production of a vitreous mullite refractory. This type of refractory had been used for several years at the Corning Glass Works.

The process consists essentially of the introduction into the top of an electric furnace of a mixture of several clays of high alumina content and melting it. The molten aluminum silicate is tapped at intervals into molds built from sand slabs. The molds containing the cast blocks are sent to storage where they are annealed for 6 to 10 days before the blocks are removed to final storage or shipment. This sequence of apparently simple operations and the present system of accurate control were preceded by the customary long period of developmental difficulties.[20]

Diaspores supply the necessary alumina addition to kaolin. Both minerals are mined within reasonable distance of the plant.

The outstanding quality of this product is its almost zero porosity, which makes it particularly resistant to corrosion, and therefore very valuable in glass house refractories, especially tank blocks. Uses for it are being found in other fields.

Service Tests and Future Possibilities of Sillimanite or Mullite Products

Lack of experience with synthetic sillimanite and the high cost of preparing it limited its use in the beginning to spark plug cores. Its use for these resulted in a real improvement in quality and warranted the expense. Other products, such as protecting tubes for pyrometer thermocouples, special furnace parts, etc., were also made to a limited extent.

The discovery of andalusite in California and the importation of cyanite from India, together with the experience gained from experimental development work, added increased interest in further development. Cyanite from the southern states and the introduction of cheaper methods of manufacturing synthetic mullite also had their share in the development. By far the greatest tonnage of the natural minerals and synthetic substitutes will be used in refractories, although some tonnage of dumortierite and andalusite is being used in spark plug cores, laboratory porcelain, pyrometer tubes and similar products.

Service tests in new fields require time and care. All manufacturers now interested in the materials maintain research and development departments and realize that their products are not "cure-alls." They are proceeding with caution and have enlisted the careful cooperation of plant engineers to study how and where to use the products to give the best results. This cooperation has resulted in the development of some

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[20] F. W. Schroeder: Electric-furnace Production of High-heat-duty Refractories *Ind. & Eng. Chem.* (1931) **23**, 124.

uses of real value to the metallurgical and other industries. Some tests have been failures, but such tests have been most constructive, for obvious reasons.

At the present time, cost is the limiting factor in the almost universal use of sillimanite wherever fireclay, silica, high-alumina or kaolin brick are used. In the super-refractory field, costs are about the same, and silli-

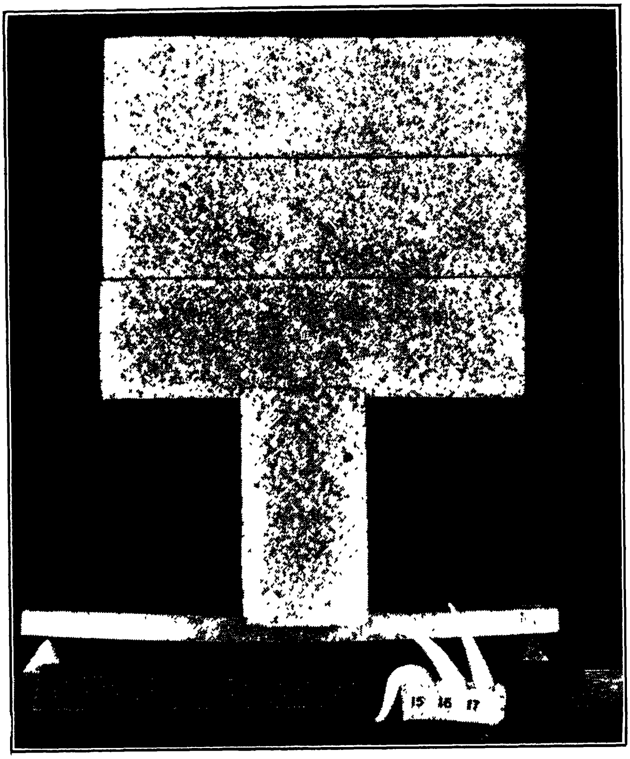

FIG. 7.—LOADED ANDALUSITE SAG BAR.

This bar, previously fired to cone 20, has held up a load of 30 lb. (13.6 kg.) or four standard 9-in. (23-cm.) bricks. The bar is 0.5 in. (12.7 mm.) thick by 2 in. (5 cm.) wide and supported by two knife edges 11 in. (27.9 cm.) apart when passed through the Dressler kiln and cone 16 melted to one-third down.

manite can compete on the basis of service alone. Recent developments indicate that processes will be perfected which will make it possible to produce sillimanite refractories at much lower prices, and permit their much wider use.

Sillimanite and mullite are giving the best results in places where refractory troubles formerly seem to have been the worst. This is so, no doubt, because these refractories, to a large extent, were developed by companies that had refractory troubles and that set to work to develop refractories to cure these troubles. They had nothing to sell, and a real incentive to produce.

The Corning Glass Works had to combat high temperatures in the manufacture of Pyrex glass and required a very dense, nonabsorbent container that corrosive glass could not penetrate. It also wanted a refractory that would improve with age. Cast mullite was the solution to the problem.

The Champion Porcelain Co. found it advantageous to work at higher temperatures than were used commercially. One refractory was available but unfortunately it deteriorated with age. Mullite produced

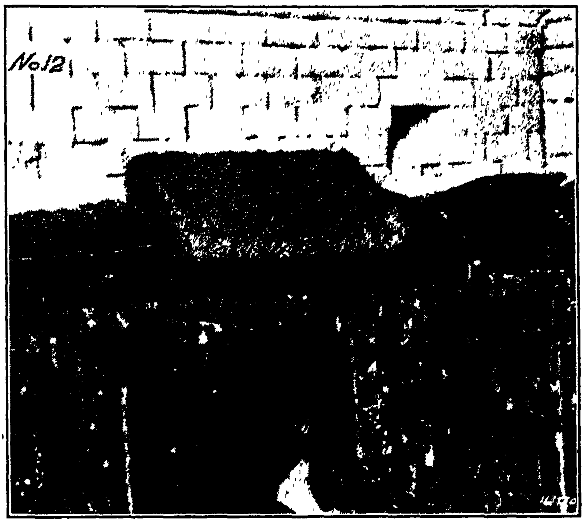

FIG. 8.—P. B. SILLIMANITE TUCKSTONE IN SQUARE FURNACE AT END OF 14 MONTHS' SERVICE.

Compare with fireclay tuckstones on either side, which originally were of the same size.

from an andalusite base solved their problem for saggers, furnace linings, and many special parts which are constantly kept at temperatures around 3000° F. (Fig. 7). Champion andalusite refractories are marketed as "Champion" sillimanite. Mullite develops and improves at these temperatures.

ELECTRIC FURNACES

It has been known for some time that if gray iron is heated to a high enough temperature (2900° F.) and agitated, the carbon can be distributed better, and gas pockets removed so as to produce extremely tough castings. These castings have tensile strengths 25 to 50 per cent greater, and transverse strengths 25 to 70 per cent greater than regular cast iron. The Detroit Electric Furnace Co. manufactures a rocking furnace constructed to give the necessary agitation of the molten metal. It depends upon andalusite refractory linings.

Electric-furnace builders have experimented with all types of sillimanite and mullite to solve their problems. They must not only have a

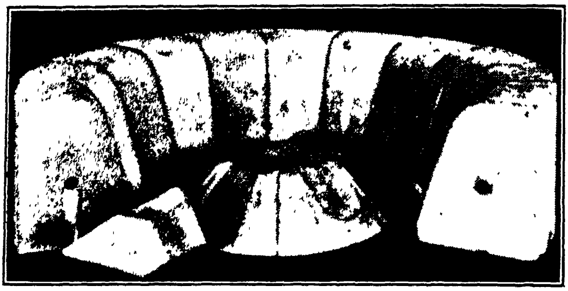

FIG 9.—POWDERED COAL BURNER CONTAINING A LARGE PERCENTAGE OF ANDALUSITE. A set of these burners was installed in six 1800-hp. boilers designed to operate at 300 to 350 per cent. The boilers are fired from one side with powdered coal and from the other with blast-furnace gas. Combustion air is preheated. The blocks containing andalusite have been in service 10 months and as yet show no serious deterioration.

FIG. 10.—COMPARISON BETWEEN CORHART CAST MULLITE BLOCKS AND ORDINARY HIGH-GRADE REFRACTORIES IN GLASS-TANK SERVICE.

good refractory that will improve in use but they must be able to make repairs in service. This is accomplished by the use of so-called ramming mixes, which are made of grains of various forms of mullite sized to

give the greatest possible density with as little plastic clay bond as possible. They can be rammed into position and heated at once. Such mixes have been made from the molten cast mullite, raw andalusite, calcined cyanite, "Argon," etc.

In several instances, sillimanite has shown 20 times the life of other refractories. Thinner furnace walls have been possible, hence increased tonnages. In one instance, a firebrick hot zone in a continuous skelp-heating furnace operating at 2900° F. had to be built completely every two weeks. Substitution of P. B. sillimanite increased the capacity and the hot zone lasted several months. Glass house refractories made from both the molten cast mullite and products fabricated from natural grain have shown increased life (Figs. 8 and 10). Conditions in various parts of glass tanks and furnaces vary, and require different treatment. Improved containers have permitted higher temperatures to increase production. This, in turn, has resulted in more severe conditions on the superstructure, hence more demand for mullite.

High temperatures are not necessarily one of the requirements. Low thermal expansion has been taken advantage of in construction of oil refineries in the West, and the increased cost of mullite has been warranted.

Even until such time as tonnages can be increased to compete in the larger tonnage fields, there is a substantial market for sillimanite and mullite and a real service saving to be performed in all sorts of high-temperature metallurgical furnaces, such as electric furnaces, forges, oil-fired furnaces, ceramic kilns, glass house equipment, etc.

Refractories of this type have a P.C.E. (Pyrometric Cone Equivalent) of cone 36 to 40, a low thermal expansion (5×10^{-6}), a good thermal conductivity as compared to clay refractories, a high softening or deforming point, and good resistance to spalling. Deformations of less than 1 per cent at 1500° C. under loads of 50 lb. per square inch are not uncommon on standard 9-in. firebrick shapes. Figs. 9 and 10 show some of the results obtained by using sillimanite and mullite under severe conditions.

Acknowledgments

The writer wishes to express his thanks to Dr. Joseph A. Jeffery for much valuable information in preparing this article; also to The Vitrefrax Corporation, The Corhart Co., and Charles Taylor Sons Co. for information regarding their products.

DISCUSSION

(W. M. Weigel presiding)

P. F. Kerr, New York, N. Y. (written discussion).—It was my privilege to visit both the andalusite mine, near Bishop, Calif., and the dumortierite mine near Oreana,

Nev., during the summer of 1931. Two of the most interesting minerals of the sillimanite group just discussed by Mr. Riddle occur in these deposits. The occurrence of andalusite associated with corundum and pyrophyllite as found in commercial quantity near Bishop is unknown elsewhere in the world. The natural mixture of minerals appears almost ideal for ceramic use. Twelve different minerals occur with the andalusite but all are either beneficial in the manufacture of porcelain or occur in such minor amounts as to be negligible.

In the view of the andalusite mine shown by Mr. Riddle in Fig. 3, about one-third of the area of exposed rock is mineralized with andalusite. The zone extends in a diagonal band from the lower left-hand corner of the view almost to the crest of the ridge on the right. The main workings are along this zone. On the opposite side of the ridge, and higher on the mountain, are other openings not visible in the picture. Also, lower down on the opposite wall of the canyon are deposits of low-grade andalusite.

The dumortierite deposit at Oreana contains a smaller number of minerals than are to be found in the andalusite mine near Bishop. It is of interest to note, however, that the dumortierite is frequently associated with andalusite and it is probable that the latter mineral adds somewhat to the value of the ore.

In Mr. Cole's sketch (Fig. 6) the dumortierite zone is roughly outlined by the distribution of the surface workings and the hard rock outcrops on the hill above the workings. The hard rock is highly impregnated with dumortierite and stands out from the soft barren sericite and tourmaline schist on either side. Another dumortierite zone of similar nature but lower grade occurs to the left of the field of view. The latter zone has not yet been mined. The dike mentioned on page 144 as encountered 160 ft. from the portal of the lower tunnel is exposed at the surface in one small outcrop just below the top of the hill. Thin sections show that the material of the dike is definitely later and is not related to the dumortierite zone.

A. B. Peck, Ann Arbor, Mich. (written discussion).—Mr. Riddle has mentioned the almost total lack of volume change in andalusite and sillimanite and the large volume change in cyanite, when these minerals break up at high temperatures to form a mixture of mullite and silica. It might be well to enlarge upon this and point out that one of the great advantages of these minerals is that, once the dissociation change has been accomplished, there are practically no subsequent volume changes of importance, regardless of the temperature ranges short of fusion and the direction of the temperature change to which the refractory may be subjected. The only variations in volume are those due to normal thermal expansion and to slight gradual shrinkage caused by chemical reaction between the constituents of the refractory, which results in the formation of additional mullite. There are no inversions of the mineral constituents with consequent volume changes such as, for example, occur in silica brick. In other words, refractories made from the minerals of the sillimanite group are monotropic, in both volume and mineralogical composition.

The fact that these minerals, with the exception of sillimanite, dissociate into mullite and silica at comparatively low temperatures is very important. First, it is possible to easily produce a very refractory mixture, and also one which because of its relatively low content of liquid (silica) does not readily deform. Furthermore, this liquid is itself very viscous and this also aids resistance to deformation.

The excess silica is an important factor in aiding refractoriness, and resistance to deformation in another way. Mr. Riddle touched upon this when he mentioned the intentional addition of corundum to the body with resulting formation of additional mullite upon long exposure to heat. The mechanics and chemistry of this are very plain if one follows the development of the microstructures of such a mixture, with the petrographic microscope.

Thus, the excess silica which is split off upon dissociation is in the form of cristobalite, but is in a very fine-grained condition, so fine as to be nearly submicroscopic or amorphous, and, therefore, in an ideal condition for further chemical reaction. The original, single-fired andalusite or cyanite brick to which corundum has been added, has, therefore, a constitution of crystalline mullite and corundum and perhaps a small amount of residual original mineral, embedded in a matrix of siliceous glass. With continued firing as in service, the mullite crystals in the original grains tend to unite and grow larger, and at the same time the corundum and siliceous glass react to form additional mullite crystals in the bond. These latter crystals grow into an intricate network, which is a great aid in producing greater strength to resist deformation. Also, it can be seen that the body is continually forming more mullite and is thus approaching that ideal state where we would have a perfectly homogeneous, one-component refractory.

A. F. Greaves-Walker, Raleigh, N. C. (written discussion).—Since the publication of the article on North Carolina cyanite referred to by Mr. Riddle (p. 140), exploration of the deposits and the development of processes for the production of high-grade cyanite concentrates has continued. Some interesting geological research on the origin of the deposits has also been carried on by Dr. J. L. Stuckey, head of the Department of Geology at North Carolina State College.

The Celo Mines, Inc., has recently completed a pilot plant near Burnsville, N. C., and is producing very pure concentrates from a disseminated deposit. All iron minerals are removed by a new electrical process.

The North Carolina mineral, of which tremendous reserves have been developed, appears particularly suited to refractory uses, especially in ramming mixes and refractory cements. Certain quantities can be used in these products without precalcination, advantage being taken of the expansion of the cyanite to overcome the natural shrinkage of other ingredients. Many authorities agree that it is only a question of time until the North Carolina cyanite becomes an important factor in certain branches of the refractories industry. There seems to be some question as to whether the North Carolina mineral will ever come into wide use in the manufacture of electrical porcelain, because of its physical characteristics. Comparatively little research has been done in this connection, however, and the possibilities are promising.

T. S. Curtis, Los Angeles, Calif.—The method of separating the quartz and cyanite which Mr. Riddle has described is for the purpose of controlling the percentage of SiO_2 in the product. The ore as mined shows 35 per cent cyanite fiber, but is not a typical cyanite in which the Al_2O_3-SiO_2 ratio is 1 to 1. The mineral from California actually shows 1.4 parts SiO_2. This permits the exudation of glass so that the fibers are welded together and strengthened. The ore in the mine consists of cyanite mixed with quartz. The quartz can be broken down by heating to the critical temperature and then quenching with water. This is achieved by having the rotary kiln discharge directly into a water bath, so that the quenching takes place within a second or two after the charge leaves the throat of the kiln. The associated quartz gangue is thus broken down and can be separated by screening.

The product is made in two standard grades, one containing 50 per cent cyanite fiber and 50 per cent free silica, the other 70 per cent cyanite to 30 per cent silica. The inversion of the free silica to tridymite is assisted by heat treatment and appropriate reagents. The product of this process has a variety of uses in low-temperature refractories, but not in vitrified materials. The refractory products of this company are based on clay deposits of California where the alumina runs as high as 63 per cent. This is brought to a state of plasticity and mullite needles of the desired size are allowed to form. Its physical stability is important, as is its low coefficient of expansion and the smoothness of the curve representing this parameter. These facts have

made possible the use of this high-grade refractory in places where ordinary refractories have commonly been used. Specifically, it is employed in oil stills which are operated at low red heats.

With reference to mullite, it can be said that this mineral represents a state of matter, the ultimate phase of stability in mixtures of alumina and silica. Attention should be called to the possibility that mullite may form even where it has not been specifically introduced.

MEMBER.—Can mullite-forming minerals replace feldspar as a refractory bond?

F. H. RIDDLE.—When the feldspar melts and bonds the clay and quartz, the body is not any stronger than the glass cement thus formed. If mullite is used, however, and the temperature carried high enough to lower the flux content to a minimum, the body is bonded with interlocking crystals, and no dependence is placed on a glass bond.

The Geology of Some Kaolins of Western Europe

By Ernest R. Lilley,* New York, N. Y.

(New York Meeting, February, 1932)

While American scientific literature contains much information upon geologic conditions controlling the production of oil in Rumania, copper in Chile, and other fuel and metallic resources in many foreign countries, it is singularly lacking in data about the nonmetallics outside of the United States. Thus, though the United States is the outstanding market for European high-quality refined kaolin, and for the chinaware, porcelain, and other products of the European ceramics industries, American literature contains only the briefest references to the deposits from which the raw kaolin utilized is obtained. To the writer, to whom has come in the past year the opportunity of visiting a number of these deposits personally, they have proved of extreme interest both commercially and geologically. It is the latter phase that is discussed in this paper.

The older and now largely crystalline rock areas of France, Germany, and other parts of western Europe contain numerous pegmatite dikes of which the upper portions have been subjected to surface weathering and kaolinization in much the same way as those of the eastern portion of the United States. Like the latter, the European pegmatite kaolins have long been known for their freedom from iron and other impurities offensive to the china and porcelain manufacturer. Unfortunately, they are also, like the American deposits, individually of small size and noted for their high costs of production.

Deposits of sedimentary clay similar in many respects to the high-quality sedimentary kaolins of Georgia and South Carolina are known and worked, notably in southern Germany and in the Newton Abbott district of Devon and Dorset in England. In general, their relationship to the parent rock from which they were derived is more evident than in the United States and the individual deposits are of smaller areal extent and less regular in outline.

While these deposits are of considerable economic importance they did not, because of their similarity to deposits in America, attract and hold the interest of the writer to the extent that several other types of deposits did. It is to the latter, which are so different from the kaolin

* Professor of Economic Geology, New York University, and John Simon Guggenheim Memorial Foundation Fellow in Europe, 1930–1931.

deposits that are being worked on a commercial scale in the United States, that he wishes to call attention. They are of three types:

1. The blanket residual kaolins in the granites and related crystalline rocks of southern Germany and Czechoslovakia.
2. The kaolinized arkosic sandstones of Czechoslovakia and Bavaria.
3. The china clay and chinastone in the granite of Cornwall and Devon, England.

Blanket Residual Kaolins of Southern Germany and Czechoslovakia

Deposits of this group reached their greatest economic importance in the German state of Saxony, the Prussian province of Saxony, and in the celebrated deposits at Zettlitz north of Carlsbad in Czechoslovakia. Similar but less important deposits are found in adjacent areas. The deposits at Zettlitz (Fig. 1) occupy a relatively small area, the eastern portion of an east-west trending graben lying between the crystalline Karlsbad Mountains on the south and the Erzgebirge on the north. Immediately north and south of the graben the outcropping rocks consist almost entirely of granites of Upper Carboniferous age. The granite also underlies the graben where it is covered by Tertiary clays, sands and lignites and in some areas by basaltic masses. The upper portion of granite has been completely kaolinized through much of the graben. The kaolinized area extends westward from the vicinity of Sodau for a distance of nearly 12 km. Its greatest breadth is approximately 4 km. Other occurrences are known, but this belt, especially the eastern half lying between Zettlitz and Sodau, is the most important both because of the quantity and the quality of the kaolin available.

While formerly the kaolin was obtained in open pits, these have given way to underground mines both on account of increasing depths and difficulties in keeping the kaolin clean in the open pits and because of regulations imposed upon the industry by the state authorities operating the Carlsbad Springs. Regulations by the same authorities, based upon the assumption that the Thermal Springs fissure, to which Carlsbad owes its existence, is directly connected with the network of water-filled fissures that cross the kaolinized area and would be adversely affected by clay mining in the southern portion of the graben, have prevented exploration of that area.

In the vicinity of Zettlitz and Ottowitz, the center of the graben, the kaolin lies at a depth of 50 to 60 m. Between the overlying Tertiary clays and the kaolin lies a thin layer composed largely of kaolin material, apparently washed out of the decomposed granite and known locally as "Josefi-Flöz," and immediately below it a layer of partly cemented sandy material, a residual breccia called "Quartzitdeckel." In areas under

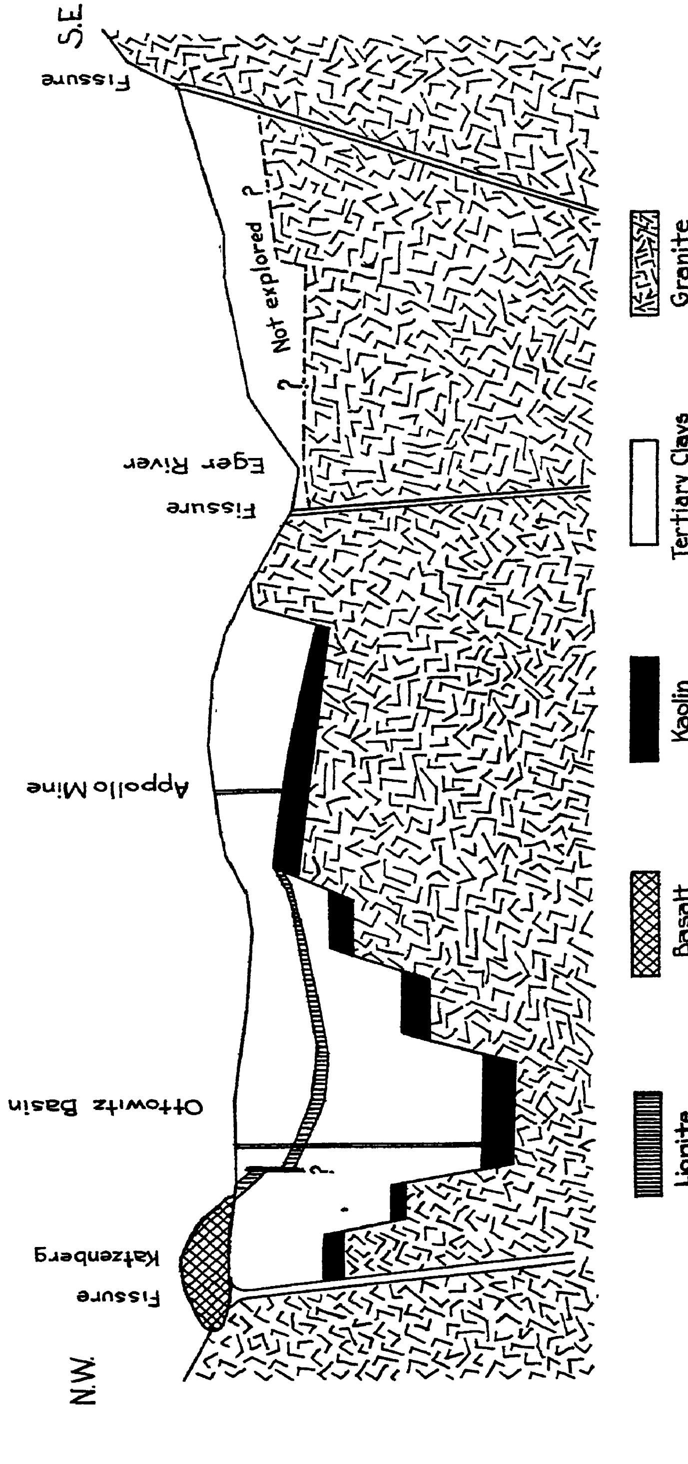

FIG. 1.—GENERAL CROSS-SECTION THROUGH THE ZETTLITZ KAOLIN DISTRICT.
Based on mine section of Zettlitzer Kaolin Works A. G.

exploitation the kaolin of sufficient purity for use is normally about 15 m. thick, although in some places it reaches 20 m. The kaolin, which lies like a blanket over the granite, grades at the base into a greenish, iron-containing, harder material which in turn grades into granite. The best quality of kaolin is usually expected in the upper and more completely kaolinized portion of the deposit. The kaolins in the several pits differ somewhat in purity, plasticity and other physical properties. The kaolin worked today contains from 25 to 40 per cent of clay substance, practically the whole of the remainder, with the exception of 1 or 2 per cent of feldspar, being quartz.

Because of the number of water-bearing fissures crossing the kaolinized area and the close proximity of the renowned Thermal Springs, there has been a tendency in the past to consider that the kaolin is the product of the alteration of the granite by the carbonic acid and related chemical matter contained in such waters. Today there seems to be a greater tendency to feel that these waters have served only as a means of modifying the process and nature of the products of kaolinization by another agency rather than as the primary cause themselves. Certainly the kaolinization is no less complete in the areas lying at considerable distances from the fissures than in the areas immediately adjacent to them. Those who are most closely connected with the exploitation of the deposits seem to feel that sulfuric acid, humic and carbonic acids, and similar matter deficient in oxygen derived from the decomposition of lignites in the overlying Tertiary, were by far the most important of the agencies causing the kaolinization of the granites. Dr. Otto Mickler, mine director of the Zettlitzer Kaolinwerke A. G., goes so far as to state that the area of kaolinization corresponds with the area that has been overlaid by lignite.

Probably the honor of being the earliest producer of China clay in Europe belongs to southern Germany, where kaolin was worked early in the eighteenth century (Fig. 2). The names of Dresden, Meissen, Aue, Wurzen, Halle, Kemmlitz, among others, have long been famous as centers of production of high-quality porcelain kaolins and of the manufacture of porcelain ware. The belt extends from the Fichtelgebirge in northern Bavaria as an arc paralleling the northern flank of the Erzgebirge eastward to the Saarau-Strehlen area in Silesia. It is natural, considering the size of the area involved, the differences in the nature of the materials subjected to kaolinization, and in the conditions under which it occurred, that the deposits should differ from each other substantially in form, extent and quality. However, the more important sections show a surprising degree of similarity in many respects; therefore the writer will limit himself to a brief discussion of two of the most important of the areas—the section around Kemmlitz lying between Meissen and Leipzig and the Halle district.

In the Kemmlitz area (Fig. 3) both open-pit and underground mines are in operation at the present time. The kaolin is found in basinlike areas in the Rochlitz Porphyry of the Rotliegendes (Lower Permian). The kaolin lies below an overburden of glacial drift and alluvium (Fig. 4), which attains a thickness of 25 m. in some parts of the basins but is almost entirely lacking along the rim areas. In the centers of the basins kaolinization probably extends to a depth of 75 m., giving a thickness of kaolin of approximately 50 m. Up to the present time only the upper portions of the kaolin in the deeper parts of the basins have been worked. Away from the centers, the thickness of kaolin decreases and unaltered porphyry outcrops between the basins, therefore it is possible to secure

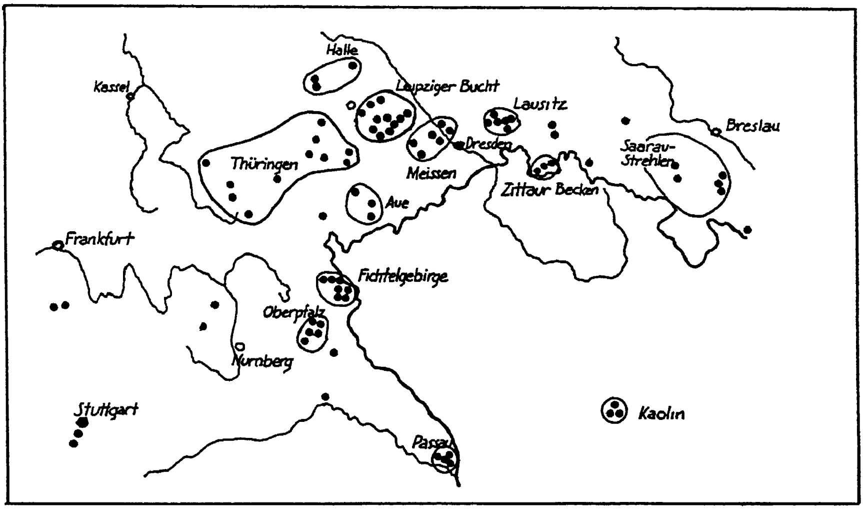

FIG. 2.—GERMAN KAOLIN AREAS. (*After N. B. Jungeblut.*)

samples giving a complete range of changes in the alteration of porphyry to kaolin of the best quality. The uppermost portions of deposits are usually somewhat discolored by material from the overlying drift and alluvium. The bulk of the deposits are white and consist almost entirely of quartz, kaolin and small amounts of feldspar. The kaolin content averages about 30 per cent, but considerable raw kaolin is mined that contains as high as 40 per cent.

Although no lignite or other carbonaceous material is found overlying the kaolin deposits today, its occurrence in areas near by indicates that it has probably been removed from this area by erosion. Thus while the evidence directly available supports the conclusion that these deposits were formed by normal weathering, most German geologists prefer the theory that these, like the deposits of Zettlitz, are also the products of alteration by acid waters derived from the decomposition of the lignites.

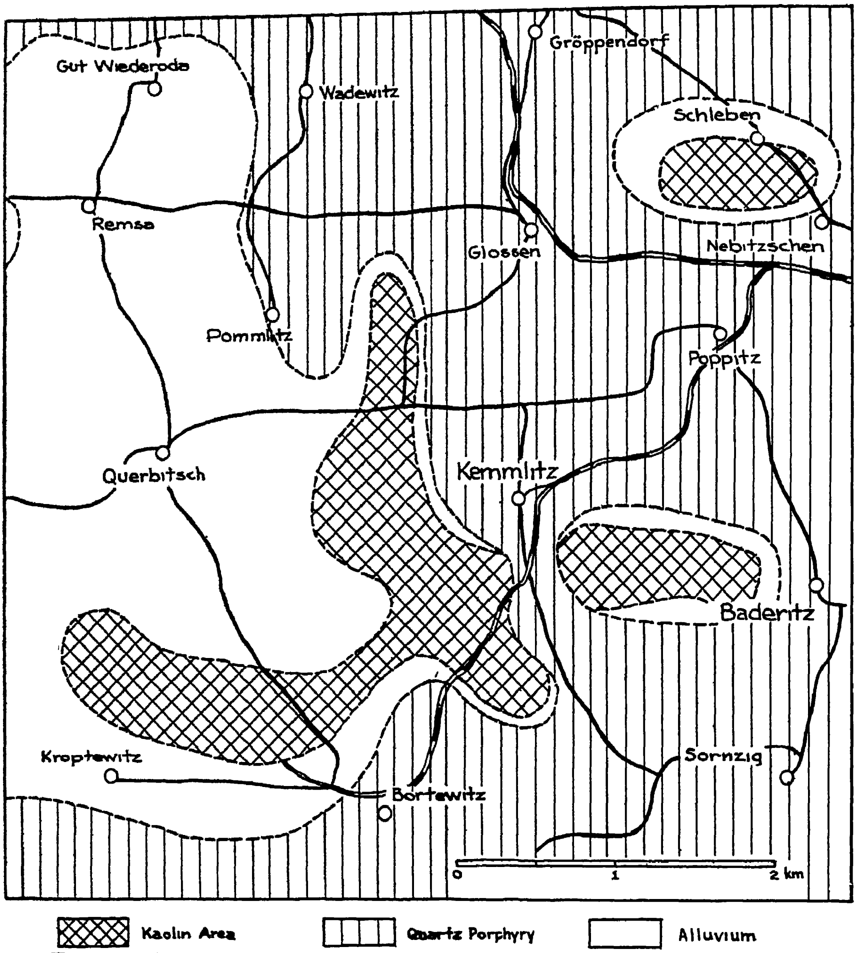

FIG. 3.—KAOLIN DEPOSITS IN THE VICINITY OF KEMMLITZ. (*After J. Behr.*)

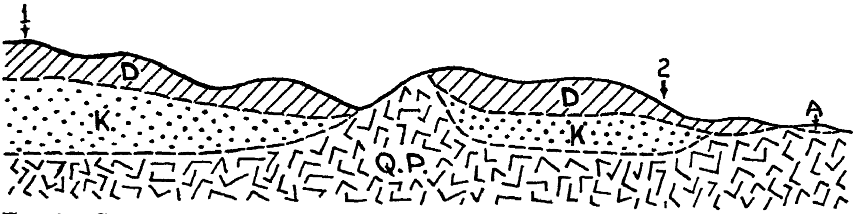

FIG. 4.—GENERALIZED SECTION THROUGH KEMMLITZ-BADERITZ AREA. (*After Stahl.*)
Length: height = 1:5
Length of section approx. 3050 meters

A. Alluvium
D. Drift
K. Kaolin
Q.P. Rochlitz quartz porphyry

1. Site of Kemmlitz shaft
2. Site of Baderitz winding shaft

The deposits of the Halle area (Figs. 5 and 6) are of a somewhat more varied nature than those of the Kemmlitz district because of differences in

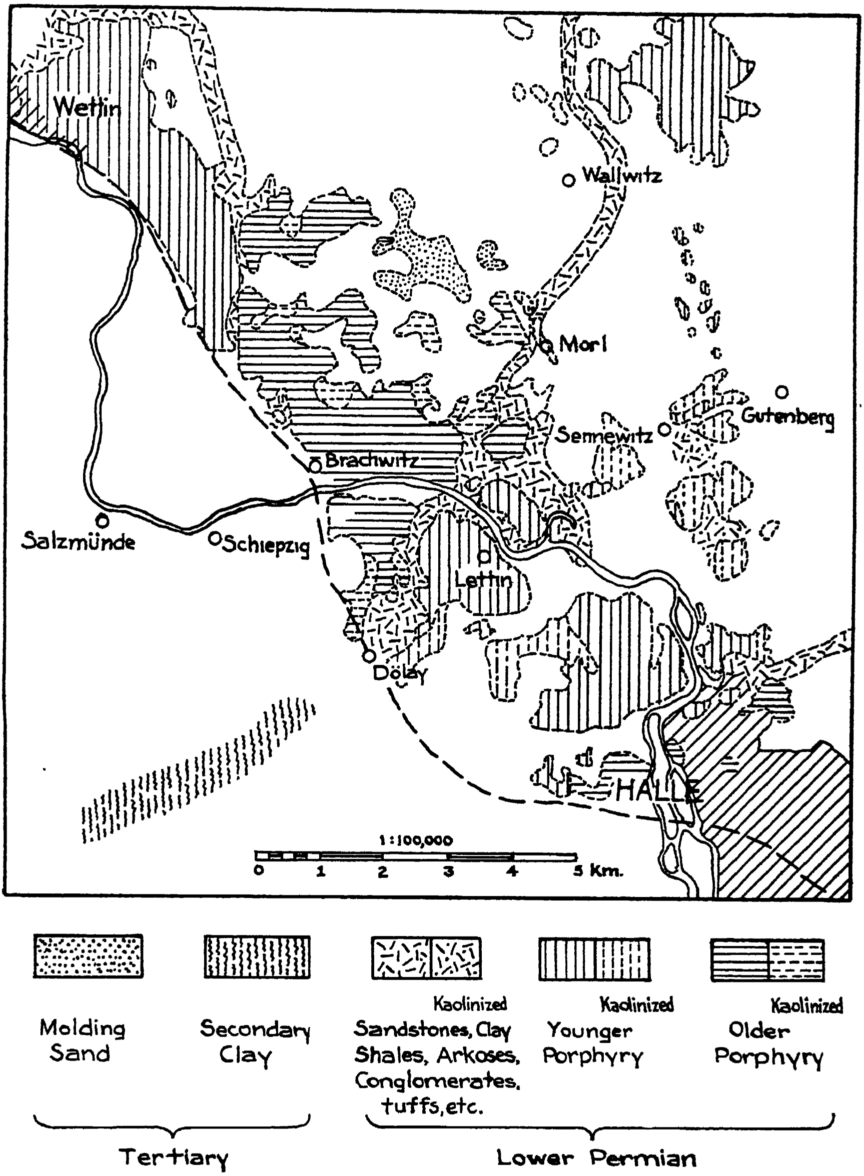

FIG. 5.—MAP OF KAOLIN, SECONDARY CLAY AND MOLDING SAND DISTRICT NEAR HALLE. (*After J. Behr.*)

the character of the rocks subjected to kaolinization. In some areas the kaolin is overlaid by a series of Tertiary sands and clays containing thin seams of lignite. In others, the lignite, though not found today, appears

to have been present until comparatively recent time. In some places practically the whole thickness of the Tertiary has been removed and the kaolin deposits are found outcropping at the surface. To the southwest of the main residual kaolin area lies an important deposit of high-quality transported clay, undoubtedly derived from the adjacent residual kaolins.

While the depth of kaolinization is generally not comparable with that at Kemmlitz, the wide areal distribution of the deposits is noteworthy. Furthermore, the agencies of alteration did not limit their activities to one type of rock. The oldest rocks outcropping in the area are known as the Older Porphyry (Lower Permian). Kaolin resulting from the altera-

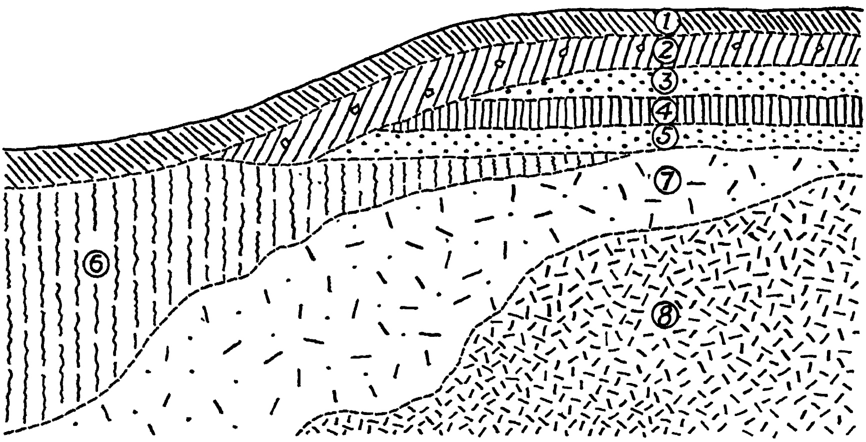

FIG. 6.—DIAGRAMMATIC CROSS-SECTION AND PART OF HALLE KAOLIN AREA. (*After J. Behr.*)

1. Soil
2. Alluvium
3. Tertiary sand
4. Tertiary clay
5. Tertiary sand
6. Kaolinized Lower Permian beds
7. Kaolinized porphyry
8. Granite porphyry

tion of this rock is worked in several places but its quality is not generally as high as that produced by kaolinization of the Younger Porphyry (Lower Permian). While the purity of this varies considerably between different pits, and even vertically and horizontally within the area of a single small pit, its high average quality has given it an excellent reputation among porcelain makers. Immediately above the Younger Porphyry and between it and the Older Porphyry lies a series of sandstones, clay shales, arkosic sandstones, conglomerates, and tuffs that have been kaolinized locally. They are, however, purely of scientific interest in this locality.

Though it is generally admitted that other agencies may have been of importance locally, the presence of beds of Tertiary lignites immediately above the purest of the kaolin deposits has made this area one of the most frequently cited by German geologists in support of the theory of the formation of such deposits by solutions derived from lignite decomposition.

Kaolinized Arkosic Sandstones of Bavaria and Czechoslovakia

Near Altenburg in Thuringia, in the Oberfälz district of Bavaria, and in the Pilsen Basin in Czechoslovakia are found examples of a most interesting group of deposits, kaolinized arkosic sandstones. Those of Thuringia are of limited importance commercially at the present time, but the deposits at Pilsen and at Oberfälz are of major economic importance, being especially in demand as a filler for papermaking and similar purposes. In Thuringia, the Bunter sandstone of the Upper Rotliegendes (Permian) age has been extensively kaolinized. The kaolin is generally considered to have been formed from feldspar contained in an arkosic sandstone by alteration *in situ* through the agency of waters descending from overlying lignites. This view is supported by the kaolinization of the upper portion of the mica porphyrite lying immediately below the sandstone in some areas. Because of its high content of iron the use of this kaolin has been greatly restricted in comparison with the use made of the similar but higher quality kaolins of Czechoslovakia and Bavaria.

The greater part of the kaolin obtained in Bavaria comes from the Oberfälz or Amberg district northeast of Amberg, with production coming from Hirschau, Schnaittenbach, Neunaigen, Kohlberg, Weiterhammer, Tanzfleck, Freihung and Steinfels (Fig. 7). The largest production comes from the area between Hirschau and Schnaittenbach. Here the kaolin containing sandstone belongs to the Middle Keuper (Triassic) and outcrops as a belt usually from 50 to 150 m. wide, but here and there as much as 300 m. wide. The belt may be traced for a distance of 9 km. to an area north of Neunaigen. Its full extent is not known, as test holes have been sunk into the deposits for only a depth of 25 m. The highest percentage of kaolin present in the sandstone is 28 per cent, but the average is lower, much of that worked near Hirschau containing only 10 per cent. In sections immediately adjacent to those being worked the kaolin content in the sandstones falls as low as 0.5 per cent. Differences in kaolin percentage are for the most part directly traceable to the degree of alteration of the pink feldspar characteristic of the arkosic sandstone of the section, the percentage of the kaolin being inversely proportional to that of the feldspar. The less completely kaolinized sections contain fresh mica, which tends to disappear in the more highly kaolinized parts. Cerussite is present in small quantities. The beds dip toward the northeast under an overburden of alluvium at the rate of about 10°. Between individual beds of kaolinized sandstone thin beds of greenish clays are frequently found. Close to the sandstones these are frequently speckled with iron oxides. Stahl and other writers state that the quality of the kaolin tends to improve with depth; also, that the reddish Triassic sandstones are free from kaolin in the areas immediately adjacent to this comparatively narrow belt of excellent quality white kaolinized sandstone.

To the north, in the vicinity of Tanzfleck, Freihung and Steinfels, similar beds, also of the Keuper formation, are worked. The sandstone in

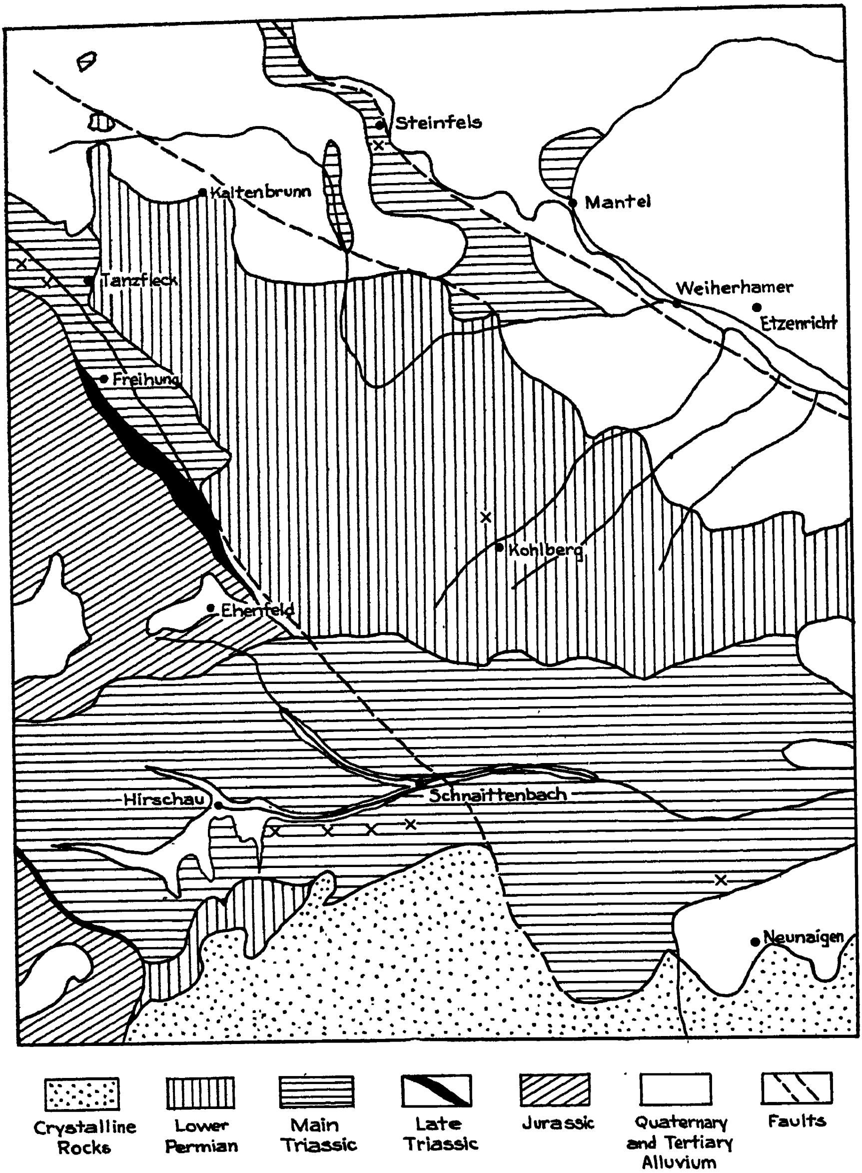

FIG. 7.—MAP OF OBERFÄLZ KAOLINIZED SANDSTONE AREA. (*After Stahl.*)

these areas, especially at Steinfels, contains only a relatively low percentage of kaolin, usually about 5 per cent. Inasmuch as the content of feldspar averages from 30 to 35 per cent and is relatively pure and free

from iron, the deposits are worked primarily for that product, the kaolin being secured as a by-product. Lead is also present in these sections.

Arkosic sandstones contained in the Permian of this area have also been kaolinized to some extent but the deposits are not as extensive nor of as high a quality as those of the Triassic and are little worked. The origin of the kaolin in the sandstones has been the subject of much controversy, because of the narrowness of the belt of kaolinization, the tendency of the kaolin to improve with depth, and the presence of lead. It has been suggested that the kaolin was formed prior to the deposition of the sandstone that now contains it and was deposited simultaneously with the quartz and feldspar with which it is now associated. The sudden and erratic changes in the ratio between kaolin and feldspar indicate that such an explanation is hardly probable. Stahl and Kohler have advanced the theory that the kaolinization has taken place *in situ* through the action of solutions rising from below along fracture planes or fissures. They differ, however, in regard to their conclusions concerning the origin and time of introduction of the lead. The writer is inclined to believe that the tendency of the kaolin to improve with depth in the relatively shallow boreholes that have been drilled in this area, together with the presence of the lead, has been given undue importance and that kaolinization through the action of downward percolating waters, possibly derived from the decomposition of organic matter of surface origin, will be found to have been of no less importance in the development of the kaolin in these arkosic sandstones than it was in the residual kaolins produced from granites and related rocks in near-by areas. From the data available at the present time, however, it would seem rather hazardous to make positive statements concerning this extremely important area.

The kaolinized arkosic sandstones of Pilsen are found in a large Upper Carboniferous basin lying to the northwest of that city. Underlying the Carboniferous is the highly folded and metamorphosed Algonkian (Fig. 8). The lower half of the Upper Carboniferous is known as the gray zone. Above this is the red zone, consisting of red sandstones and clays, with some lime, and with considerable thicknesses of beds of distinctly arkosic character. Some of these beds have been extensively kaolinized. The excellence of the quality of the kaolin for use in papermaking, and the ease with which it can be produced, especially by the open-pit method at Horni Briza (Oberbris), have made the area famous throughout the continent of Europe.

Above the sandstones lie remnants of what was probably a rather extensive but comparatively thin covering of Tertiary clays and sands with more or less lignite.

Analysis of the kaolinized sandstones indicates that they will contain between 25 and 40 per cent of kaolin, the remainder being almost entirely quartz. They contain a minimum of potassium, sodium, calcium and

magnesium, and relatively little iron. However, the composition may change within a short distance laterally to that of a typical arkosic sandstone showing little or no kaolin and a high percentage of unaltered feldspar. Individual beds of the kaolinized sandstone seldom are more than 20 m. thick, but at Horni Briza a thickness of 70 m. (including a small amount of nonkaolin material) is found. Between individual beds layers of white, red and variegated clays with concretions of limonite and similar material frequently occur. The overburden consists of more or less altered arkoses and alluvial loams and clays.

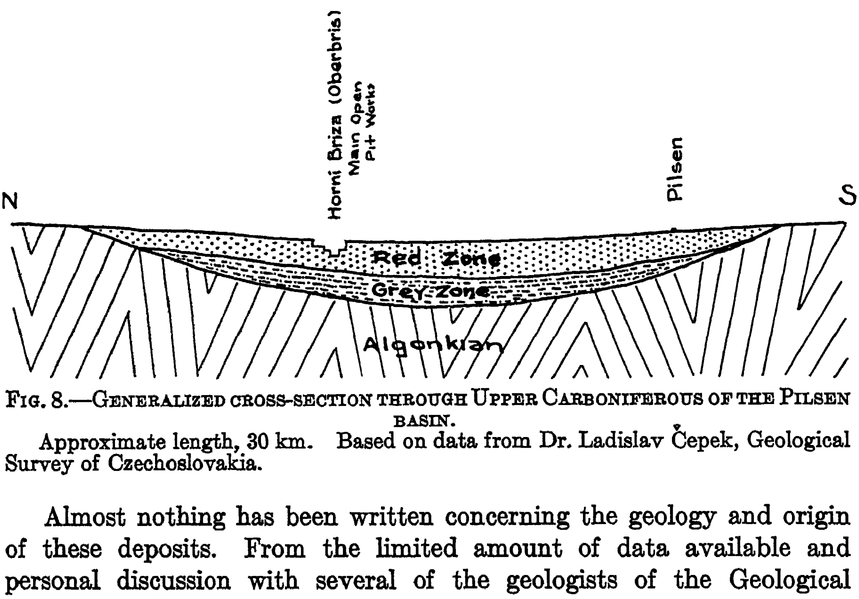

FIG. 8.—GENERALIZED CROSS-SECTION THROUGH UPPER CARBONIFEROUS OF THE PILSEN BASIN.

Approximate length, 30 km. Based on data from Dr. Ladislav Čepek, Geological Survey of Czechoslovakia.

Almost nothing has been written concerning the geology and origin of these deposits. From the limited amount of data available and personal discussion with several of the geologists of the Geological Survey of Czechoslovakia, the writer concludes that the most logical explanation of the occurrence of kaolin in these sandstones is that of alteration of the feldspar content by downward percolating and circulating waters derived from the surface. It would appear probable that the greater part of the kaolinization took place during the Tertiary, and that the solutions responsible for the alteration were largely derived from the decomposition of the peats and lignites developed during that period.

CHINA CLAY AND CHINASTONE OF CORNWALL AND DEVON

The china clay and stone of Cornwall and Devon occur as an alteration product of the granite masses of which bold outcrops feature the topography of that area. The largest of these are known as the Lands End, Penryn, St. Austell, Bodmin Moor and Dartmoor masses. The last is in Devon, while the others lie in Cornwall. All have been subjected to kaolinization to some degree but the largest and best deposits and the

most extensive operations are those in the St. Austell and Bodmin Moor masses and in the southwestern portion of the Dartmoor masses (Fig. 9).

While the granite shows many variations of scientific interest, it may be generally described as a gray porphyritic granite in which feldspar crystals are prominent. Orthoclase crystals veined with albite compose the bulk of the feldspar. Some soda-rich plagioclase is also present. Large quartz crystals are not infrequent but usually less common and less regular in form than the feldspar crystals. The matrix contains biotite, muscovite, albite, orthoclase and quartz, with varying amounts of tourmaline. Apatite, zircon, magnetite, cordierite, topaz, andalusite and fluorspar are frequent as accessory minerals, the frequency varying

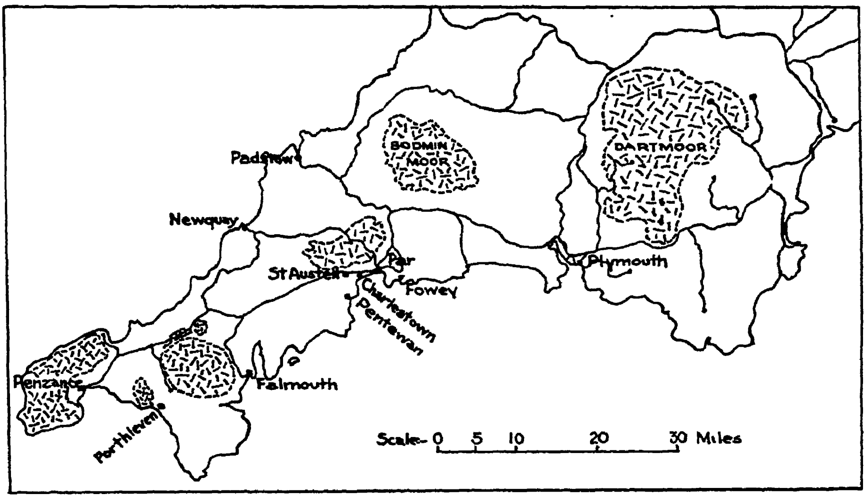

FIG. 9.—MAP OF CORNWALL AND PART OF DEVON, SHOWING LOCATION OF GRANITE MASSES. (*After J. A. Howe.*)

in different areas, that of the topaz being especially high in the St. Austell area. While the tourmaline occurs frequently as a late development formed at the expense of the biotite, the greater part is believed to be of primary origin and indicative of the presence of unusually large amounts of boron and hydrogen in the original magma. In some places, as near St. Stephen in the St. Austell area, the fluorspar is particularly abundant and is believed to indicate an abundance of fluorine in the post-consolidation and pneumatolytic stage in the development of the granite (Fig. 10).

The kaolinized areas in the granite must not be thought of as of limited extent. The outlines of most of the larger areas, after many years of mining, are known only in the most general way, the area of exploration and development work being restricted to ascertaining the existence of a sufficiency of high-quality material to warrant the construction of the necessary washing and drying units. Practically no

real attempts have been made to ascertain the depth of the deposits because of the impossibility under present competitive conditions of working deposits that are too deep for open-pit operation.

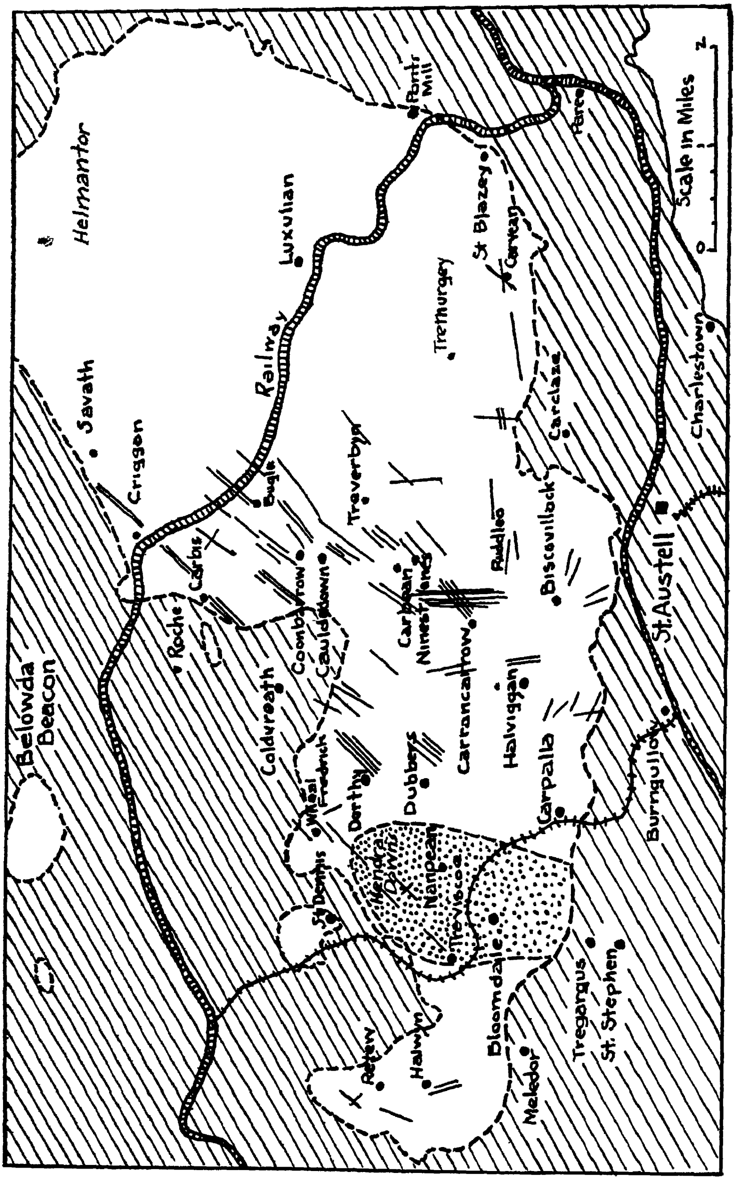

FIG. 10.—MAP OF ST. AUSTELL DISTRICT. (*After J. A. Howe.*) Straight lines represent tin veins. Granite area largely kaolinized, especially near tin veins. Speckled area represents chinastone.

The relatively low resistance to weathering offered by the kaolinized areas as compared by that offered by the unaltered granites surrounding them has resulted in the latter forming the higher ridges, while the kaolin

underlies the valleys between them. The workable kaolin commonly lies below a relatively thin overburden of soil with more or less peaty material and discolored, usually yellow, kaolin. The transition from overburden to relatively pure kaolin is usually rather abrupt and sharply defined. The kaolinized area may be so small as to be commercially valueless or it may be, as in the Stannon Marsh area in Bodmin Moor, a belt more than a mile long and one-half mile wide. It is customary to give but slight attention to the question of the depth of kaolinization, practice having shown in practically every case that the depth to which kaolinization extends exceeds the maximum depth to which the pits may be carried at a profit. While operations have been carried to a depth of 300 ft. in some pits, not only was there no indication of decrease in the degree of kaolinization with depth but, in some cases at least, there is reason to believe that the process has been more thorough and complete at the lower depths than it was close to the surface. As no boreholes appear to have been drilled below the bottom of the pits, it is not possible to estimate the depths at which kaolinization ceases.

The raw kaolin usually contains from 12 to 30 per cent of clay substance, although a content of as high as 50 per cent has been recorded. The average content of the kaolin worked today lies between 20 and 25 per cent. The remainder, called sand, consists mainly of quartz, black tourmaline, gilbertite and white mica, with some topaz and fluorite. Comparison of analyses of typical china clays and unaltered granites (Table 1) shows them to differ chemically mainly in that the percentage of the alkalines is materially lower than in the granite. The water content is naturally higher. The iron content of the raw clay appears to average more than 1 per cent, but this is of little commercial importance because the larger part is removed in the washing. The degree of cohesion of the clay is low, lumps crumbling readily in the hand and the sides of the pit disintegrating rapidly under the force of the hydraulic jet.

The great depth and wide extent of these remarkable deposits impose great difficulties upon those attempting to explain their origin. The problem is further complicated by the fact that the granite has been subjected to other types of alteration which most students of the area consider to be genetically related to the process of kaolinization. It is therefore necessary that these be considered in some detail.

Subsequent to the solidification of the granite and of the numerous intrusive quartz-porphyry dikes that it contains, the area seems to have been invaded by pneumatolytic solutions and vapors that have given rise to veins of schorl rock (tourmaline granite), greisen and quartz, containing more or less cassiterite. The veins have long been worked in some areas for their tin content and for the tungsten and other metallic elements that are not infrequently associated with it. A common form

TABLE 1.—*Comparison of Analysis of Typical Specimens of Granite, Chinastone and China Clay from Cornwall*[a]

	Granite[b]	Chinastone[c]	China Clay[d]
SiO_2	70.17	72.28	71.15
TiO_2	0.41	0.05	0.16
Al_2O_3	15.07	14.90	19.41
Fe_2O_3	0.88	0.50	1.32
FeO	1.79		0.09
MnO	0.12	0.01	0.09
CaO	1.13	1.66	0.21
MgO	1.11	0.15	0.45
K_2O	5.73	5.25	1.44
Na_2O	2.69	3.01	0.05
Li_2O	0.11	0.02	0.03
H_2O at 105° C	0.18	0.13	0.16
H_2O above 105° C	0.70	0.68	5.09
P_2O_5	0.34	0.53	0.07
Cl	0.06	0.02	Trace
F	0.15	0.88	0.11
S	0.04	Not found	Not found
B_2O_3	Strong trace	Not found	0.33
ZrO_2		Trace ?	
	100.68	100.07	100.16
Less O for F	0.07	0.37	0.04
Total	100.61	99.70	100.12

[a] Analyses by Dr. W. Pollard, from Handbook of Kaolin, China Clay and Chinastone, by J. Allen Howe.

[b] From Lamorna quarry, Cornwall.

[c] "Hard purple" chinastone from Goonvean, near St. Stephen, Cornwall.

[d] China clay rock from Georgia works, Cornwall.

of alteration of the granite resulting from the pneumatolytic activity is that of tourmalinization, tourmaline and quartz replacing feldspar and mica. A second form frequently present is greisenization, in which white mica and quartz are developed at the expense of the feldspar.

Probably the most important, both from a commercial and a scientific standpoint, is the development of chinastone—a peculiar form of alteration or differentiation in the granite in which the orthoclase feldspar normally present is largely replaced by plagioclase feldspar, generally oligoclase, and the biotite mica by muscovite or its special forms, sericite and gilbertite. The quartz is generally smoky and is in sharp contrast with the white feldspar. Topaz and fluorspar are characteristic minerals; magnetite and tourmaline while frequent are less abundant. If the latter is present in amounts higher than usual, the rock is usually much decomposed and kaolinized.

The type locality for this rock is the area around St. Stephens (Fig. 11) and northward in the St. Austell granite mass, where deposits covering a number of acres and of unknown vertical extent are being worked. A somewhat similar material, but one that contains so much dark mica and ferruginous material as to render it useless for pottery work, is found in the Dartmoor mass.

Some confusion seems to have developed in the early literature concerning these deposits because of the assumption that the partly weathered "white form," very abundant at shallow depths, was the normal type. This has been found by deeper operations to grade vertically into a holocrystalline rock of purple cast, the true chinastone. The following section from a typical quarry, given by J. M. Coon, indicates the transition.

Vegetable mold	Surface to 2 ft.
"Growan," a stained and disintegrated granite	2 to 12 ft.
"Buff Stone" (iron-stained) and friable "White Stone"	12 to 30 ft.
Compact "Dry White"	30 to 50 ft.
"Mild White" and "Mild Purple"	50 to 70 ft.
"Hard White" and "Hard Purple"	70 ft. to bottom of pit (120 ft.)

In the St. Stephens area normal granite, chinastone and highly kaolinized granite are found in close proximity to each other. The partly weathered white chinastone shows considerable development of kaolin but the purple rock appears to be only slightly kaolinized. The vertical depth of the kaolinization of the chinastone is decidedly limited in all areas except a few that appear to be characterized by extremely high percentages of tourmaline. The shallowness of the kaolinization in the majority of the chinastone areas suggests that the causal factor was some agency of surface derivation. One might go further and suggest that the same agency was probably responsible for the weathered and somewhat kaolinized shallow, valueless material that is found at the upper surface of the granite in various parts of Cornwall. However, such incomplete and shallow alteration contrasts so sharply with the completeness and great depth of the alteration that has made the china-clay pits of Cornwall and Devon famous all over the world that it is impossible to believe that they could have been brought about by the same agency.

Among those who have studied the area most carefully there appears to be no doubt that the alteration of granite leading to the development of the kaolin deposits was from below, and by some agency closely connected with the latter stages of the igneous activity, which resulted in the formation of the granite masses themselves. In other words, the deposits were the product of some form of pneumatolytic action. It does not appear probable that the deposits were formed at the same time as the

chinastone; it would probably be more logical to consider that the chinastone was largely a true product of differentiation in the solidifying granitic magma and that, except as it indicates the existence of a magma of unusual composition, it has no bearing upon the formation of the kaolin itself.

That the abnormal composition of the magma resulted in the formation of a residual solution of even more abnormal a composition seems fully evidenced by the remarkable development of greisen, schorl rock, stanniferous quartz veins and related phenomena that have made Cornwall a paradise for the mineralogist and the geologist. That these solutions have brought about the kaolinization as well seems to be generally accepted by the geologists who have studied the area. The increase in the completeness of alteration with depth, the close proximity of kaolinized material and veins and lodes, and the abundance of minerals characteristic of pneumatolytic zones in the kaolin, have all been cited in support of this conclusion. Considerable work has been done with the microscope and much evidence has been accumulated to indicate that both fluorine and tourmaline took an active part in the kaolinization. However, it would appear that much doubt still remains concerning the exact functions of the different ingredients of the pneumatolytic solutions.

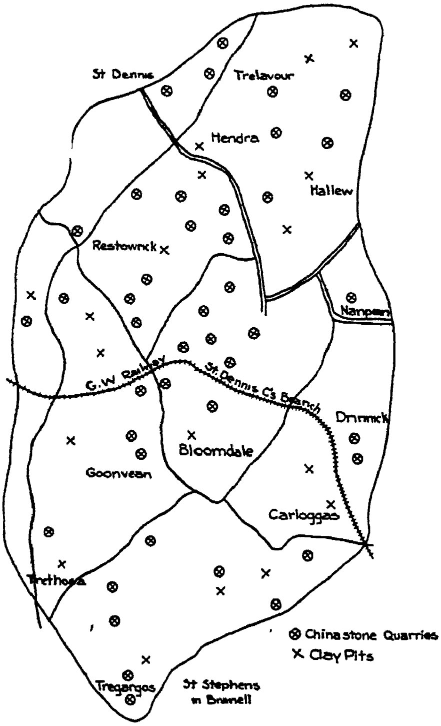

Fig. 11.—Sketch map showing distribution of chinastone in parishes of St. Dennis and St. Stephens. (*After J. M. Coon.*) Limits of area subject to revision.

Summary

The writer has endeavored in a brief way to describe three groups of kaolin deposits which were of unusual interest to him because of their difference from deposits now being worked in the United States and concerning which very little has appeared in American literature. In discussing the origin of these deposits he has of necessity discussed sub-

TABLE 2.—*Analyses of Typical European Kaolins*[a]

Locality	Composition, Per Cent										
	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	K_2O	Na_2O	Loss on Ignition	Quartz	Feldspar	Clay Substance
Czechoslovakia											
Zettlitz—washed	46.09	39.28	0.76	0.15		0.12	0.03	13.58	1.2	0.4	98.4
Sodau—washed	46.23	39.11	0.67	trace		0.21		13.78	0.9	0.2	98.9
Pilsen—raw	85.60	8.85	0.70	0.78	0.19	0.62		3.58			
Pilsen—washed	48.95	36.01	0.74	0.91	trace	0.75		12.87			
Oberbris and Wobora—raw	70.21	21.75	0.56	0.29	0.07	1.38		6.33			
Germany											
Kemmlitz—washed	58.3	29.31	0.87	0.51	trace	1.26		10.62	16.4	2.9	80.7
Lettin (Halle)—raw	57.08	29.94	0.65		0.49	2.26		9.87	17.21	8.70	74.09
Lettin (Halle)—washed	53.29	32.80	1.13	0.31	0.20	0.72		11.54	11.65	0.95	87.40
Halle—raw									34.8	1.5	63.7
Hirschau (Oberfälz)—washed	47.84	38.19	0.70	0.65	trace	0.66		12.20	1.75	3.00	95.25
Amberg (Oberfälz)—raw	84.85	8.42	0.83						75.67 per cent sand		
England											
Cornish—commercial washed	46.17	38.42	0.43	0.09	0.04	2.77		12.01	2.20	0.62	97.18
Lee Moor Pit (Dartmoor) washed	47.10	39.42	0.23	0.31	0.24	0.16	0.08	12.24			

[a] Analyses from B. Dammer and O. Tietze: Die Nutzbaren Mineralien mit ausnahme der Erze und Kohlen, 1928.
J. A. Howe: Handbook of Kaolin, China-clay, and China-stone, 1914.
B. Kerl: Handbuch der Gesamten Tonwarenindustrie, 1907.
F. Singer: Die Keramik im Dienste von Industries und Volkwirtschaft, 1923.

jects concerning which there is still more or less controversy. He has limited himself largely to presentation of conclusions that have received general approval and to which he himself could subscribe. He feels that any comparison of the commercial importance of these deposits would be out of place in this paper; each one is of especial importance in its own sphere. However, he would like to bring out one point, a query rather than a conclusion. The differences between the deposits discussed and those worked in the United States, both as regards form and character and as regards origin, lead him to ask whether the absence of such deposits in this country is not more apparent than real and to wonder if in the light of European experience this country has really made a thorough survey of its possibilities as a producer of high-grade kaolins.

ACKNOWLEDGMENTS

The writer wishes to acknowledge his indebtedness to all of those who have so courteously given of their time and knowledge. Among these he would mention particularly Prof. E. H. Davison of Camborne,

Cornwall; Mr. J. M. Coon of St. Austell, Cornwall; Dr. J. Behr of the Geological Survey of Prussia; Dr. Franz Bley of Halle, and Dr. L. Čepek of the Geological Survey of Czechoslovakia. He wishes also specifically to acknowledge his indebtedness to the officers and staffs of the various companies who arranged for him to inspect their pits and mines.

Much of the material used by the author in preparation of the paper has been published for local circulation only and is not generally available. Of the more readily available material he has found the following especially valuable and would recommend the following references to any who may desire more detailed data on particular areas:

J. M. Coon: On China Stone, Cornish Stone, or Petunzyte, *Trans.* Roy. Geol. Soc. of Cornwall (1913) **13**, Pt. 9.

E. H. Davison: Handbook of Cornish Geology, 1926.

B. Dammer and O. Tietze: Die Nutzbaren Mineralien mit ausnahme der Erze und Kohlen, 1928.

J. A. Howe: Handbook of Kaolin, China-clay, and China-stone, 1914. A publication of the Geological Survey of Great Britain.

A. Stahl: Die Verbreitung der Kaolinlagerstätten in Deutschland, 1912. A publication of the Geological Survey of Prussia.

Prospecting for Gold in the Shield Areas of Canada, Siberia, Southern Rhodesia and Western Australia

By W. H. Emmons,* Minneapolis, Minn.

(New York Meeting, February, 1932)

Although gold is one of the rarer metals, it is widely distributed; it is found on all of the continents and in each of the grand metallogenic provinces of the earth. It is prominent particularly in the shield areas of the earth; these areas have produced a large part of the world's lode gold and in most of them gold lodes are more important than the lodes of other metals. The Scandinavian shield is the only one that has not produced much gold.

Nearly all of the world's gold has been derived from lodes and from sands and gravels where gold placers formed by the destruction of lodes. The gold derived from magmatic segregations, from pegmatites and from contact metamorphic deposits is quite subordinate to that derived from auriferous lodes. The lodes include all veins and related deposits that are formed by hot waters that move along fissures or other openings and that deposit vein matter in the fissures and by replacement of the country rocks along the fissures.

All of the shield areas of the earth are characterized by large areas of granitic rocks including granitic gneisses, granodiorites and other acid intrusives which here are included in the general term "granite." The granites are intruded into schists, ancient lavas and other rocks which here are designated as the "invaded" rocks. The granites are parts of what once were great batholiths: that is, they are the remaining portions of great deep-seated bodies that slope outward, have very irregular roofs and so far as known have no floors but extend downward to great though unknown depths. Some of them are scores or hundreds of miles across and contain islands or "roof pendants" of the invaded rocks at places where the roofs of the batholiths were low—that is, where erosion has not yet completely unroofed the batholiths.

It is in these islands of older rocks and in the narrow marginal belts and cupolas of the batholiths that practically all of the gold lode deposits of the shields are found. Valuable lodes are lacking generally in the granites themselves except in granite within a mile or less from the contacts with rocks of the islands (Fig. 1). It is probable that the great areas of granites and gneisses that surround the islands are not all of the same age, and that at places unmapped granite intrudes gneiss or older

* Geologist, University of Minnesota.

granites, yet it is a fact that important areas of gold lodes are practically wanting in the granites far from the roof pendants of the invaded rock, except at a few places where the "younger granite" is recognized and described.

Not all places in the islands of older rocks are equally favorable places for prospecting. The majority of the greatest gold districts of the shields are well within the islands and the majority of the most productive districts are near small stock-like intrusives of the granites or of porphyries that are enclosed in the invaded rocks (Fig. 2) or near fingers of the granitic rocks that extend inward from the larger masses into the invaded rocks. The most valuable deposits, moreover, seem to be

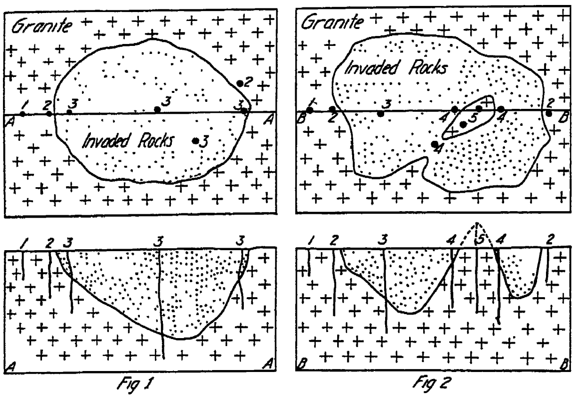

FIG. 1.—MAP AND SECTION OF ISLAND OR ROOF PENDANT OF INVADED ROCKS SURROUNDED BY INVADING GRANITIC ROCKS.

Ores are found in and near invaded rocks. A few relatively unimportant deposits are found at 1. Valuable deposits are at 2, within one mile of invaded rocks. The larger number of most productive districts are at points 3 in island.

FIG. 2.—SAME AS FIG. 1, ENCLOSING SMALL INTRUSIVE MASS OF GRANITIC ROCKS.

Ores are found in and near invaded rocks but are greatly concentrated in and around small intrusion of invading rocks that is surrounded by invaded rocks. Greatest gold lodes of shields are at 4 and 5 in and around small intrusives (cupolas).

located around the smaller stocklike intrusives, whereas the larger ones are generally associated with less productive lodes.

One may arrange the regions of the shields in classes according to their positions with respect to granitic intrusives. In the following list these regions are arranged in order according to their productiveness, each group being followed by a more productive one. This list is the result of a statistical study of essentially all gold deposits that are well known in the shield areas named.

GOLD-BEARING REGIONS OF THE SHIELDS, ARRANGED IN ORDER OF PRODUCTIVENESS

1. Granite areas one mile or more from a contact with invaded rocks. Lodes numerous, but generally not workable. Few deposits are produc-

tive. A few small deposits in Southern Rhodesia and in Siberia. Probably some placer gold derived from such lodes in Siberian shield.

2. Granite less than one mile from contact. A few valuable lodes have been worked but most of them are small and only moderately productive compared with lodes of following groups. Owl, Acorn, Battlefields mines and other productive mines in Southern Rhodesia. A few small deposits in Canada, in Siberia and in Western Australia.

3. Invaded rocks of the "island" areas. Contain numerous deposits many of which have been profitably worked.

4. Areas of the invaded rocks near small stocklike intrusives. Conain most of the largest gold lode mining districts in the shields.

Relations of Gold Lodes in the Shields to Intruding Granitic Rocks

Throughout the earth, lode ores of the metals are in the main associated with igneous intrusives and most of them are associated with granitic intrusives. They are formed mainly on the roofs of batholiths or where there are reasons to suppose that batholiths exist, but are so deeply buried that they have not been exposed by erosion. The roofs of batholiths, as already stated, slope outward and are highly irregular in contour; irregular upward projections, ridges and cupolas mark their high points and between these are low troughs (Fig. 3).

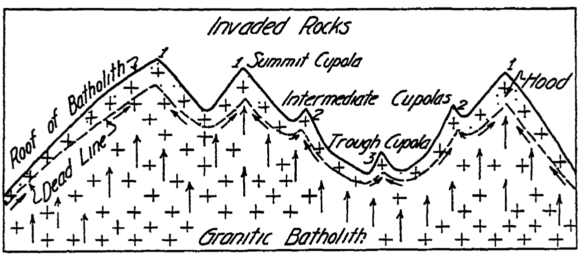

FIG. 3.—CROSS-SECTION OF IDEAL GRANITIC BATHOLITH SHOWING (1) SUMMIT CUPOLAS, (2) INTERMEDIATE CUPOLAS AND (3) TROUGH CUPOLAS.

Metallizing solutions press upward from deeper parts of batholith and converge around cupolas where a large part of the ores are deposited. Except near roof the granite itself is barren. Valuble deposits are rare in batholith more than about three miles or a little more from contact, in high regions of roof, and about one mile from low regions of roof.

Ores are found in all positions in the high parts of the roofs but the metals other than gold are found in the greatest abundance in and around the high parts. Gold segregates in largest amounts in the troughs or in small upward bulges that lie near the bottoms of the troughs. It is noteworthy also that all zoned systems of lode ores are found at the high cupolas and in belts along the sides of the batholiths, and that no zoned systems of lodes are found in the troughs or in the small low cupolas that

extend upward from near the bottoms of the troughs. No clearly defined zoned systems with gold lodes are known in the pre-Cambrian shields of Canada, Siberia, Southern Rhodesia or Western Australia.

The ores in granite that are high in the roofs lie farther from the contacts than the ores deep in the troughs. The "dead line" (Fig. 3) below which few valuable deposits are found is farther from the roof at the high cupolas than in the troughs.

These relations are evident as a result of statistical studies of essentially all mineralized batholiths. To explain them we pass from the realm of observation to that of speculation. It is believed that the mineral-bearing fluids rise to the tops of the cooling batholiths where the rock is not completely solid. They move more freely in the partly fluid batholith than in the solid roof rocks, except when fissures are provided in the roof. They tend to move to the highest points of the roofs and to concentrate around and above the cupolas, although to a lesser extent they are expressed at relatively low points also.

That is true for all metals except gold, which with much copper and zinc is deposited in largest amounts in the troughs and particularly in and near small cupolas that rise from the troughs. The central parts of the batholiths evidently are too hot for much deposition of gold, or else the metals have been expressed from the cooling igneous mass before the central or deep-seated areas solidified. The latter hypothesis appears probable, for most batholiths in their central and lower zones contain numerous strong veins of quartz or of quartz and pyrite. Such veins, however, are almost invariably barren or contain only small amounts of valuable metals. No great gold mine in the shields is in granite far from the contacts of granite and roof pendants.

The more valuable deposits are not formed below the "dead line" (Fig. 3), although the granite below the dead line is the main source of the solutions that feed the fissures in the region above the dead line. Whether this hypothesis is the true explanation or not, the relations outlined are true and should constitute a working theory for prospecting the auriferous areas of the shields. To be sure, gold ores have not been found in or around all cupolas that lie in roof pendants. Some of these may contain undiscovered deposits whereas others may be barren. The character of roof host rock, the conditions of fracturing and many other factors enter the problem, as well as the courses that were taken by the solutions that deposited the ores. All conditions must be favorable in the centers where the metals are most highly concentrated.

Preferred Positions for Gold Deposits in and near Roof Pendants

There are many productive gold lodes in the larger roof pendants of the invaded rocks where no evidences of a stocklike intrusion are dis-

covered and also in the intruding rocks near the contacts with the invaded rocks where no irregularity of the contact is apparent. In general, however, the largest deposits in the "islands" are found in relatively small areas in the islands and often where there is evidence of an underlying trough cupola (Fig. 4). This evidence may consist of (1) a small cluster of closely spaced granitic porphyry, aplite or other dikes, (2) a finger of the invading rock pointing toward the gold-bearing area, or (3) an outcrop of the cupola itself. A cluster of closely spaced granitic or porphyry dikes suggests that a granitic cupola lies below and has not yet been exhumed by erosion. A finger from the great invading granitic

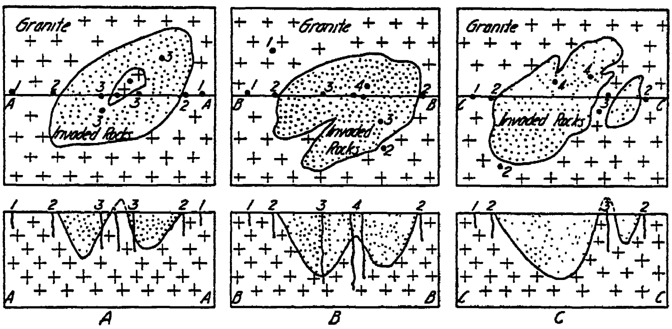

FIG. 4.—PREFERRED POSITIONS FOR LOCATIONS OF GOLD LODES.

In *A*, most valuable lodes are at 3 in and around small porphyry or granitic intrusive in large area of invaded rocks. In *B*, most valuable ores were formed at 4 above trough cupola that is still concealed. A finger of surrounding granite mass points to principal group of deposits. In *C*, deposits are found at 1, 2, 3 and 4. Deposits at 3 are low in a cupola or a ridge that has been deeply eroded and joined to main invading mass. In each diagram positions are numbered in increasing order of productiveness of deposits found at such positions.

mass suggests that the finger extends on strike below the place where its outcrop ends—that the finger is part of a granitic ridge, which extends into the greater roof pendant, and that the ridge is buried below the place where its outcrop ends. Clusters of dikes or small stocks are likely to crop out in the area of the extension of the finger and these may be interpreted as marking areas above the upward swells of the buried intrusive ridge. Outcrops of relatively small stocklike intrusives are found in many roof pendants and it is around these that many of the greatest deposits of lode ores in the shield areas are found.

The great gold-bearing districts that appear to be located in and near trough cupolas include Porcupine, Dome Lake and Kirkland Lake in Canada, and among the smaller ones are Sultana, Stabell, Red Lake, the East Ontario gold-arsenic district and many others. If copper-gold districts are included, the deposits of Noranda (Horne mine), Sherrit-Gordon and others should be listed. In Southern Rhodesia there are a score of important gold mines in the invaded rock and many are in areas

of closely spaced granitic dikes. The Shamva, which for many years was the most productive gold mine in Southern Rhodesia, is in a great roof pendant near a small trough cupola of the granite. The Kolar gold belt in India (Champion lode) is near the end of a small intruding mass of the "Peninsular" granite. The mines of Kalgoorlie in Western Australia lie on the strike of an elongated intrusive which is probably a cupola but which may be a "finger" extending into the roof pendant from the surrounding invaded granite. At Moonta, in South Australia, a small cupola contains many parallel veins. The ore is chiefly copper ore but the district is mentioned here as an example of a mineralized elongated cupola. Northeast of the end of its outcrop are the valuable copper mines of Wallaroo. The Mother Lode of California is situated at the north end of a broad fingerlike projection of the Sierra Nevada batholith. The Mother Lode region is not a shield area but belongs to a group of gold-bearing regions where erosion has not extended so deeply as in the shield areas treated here. In these the concentrations of gold near cupolas are equally marked, but such deposits do not fall within the scope of this discussion. The third diagram of Fig. 4 represents a small roof pendant near a large one. The two are separated by the invading granite. The most favored positions in the region of contact are those where there are small roof pendants in the granite mass near the contact with the main mass of the invaded rocks.

In the Battlefields district, Southern Rhodesia, the main gold lodes are in granite near a small mass of the invaded rock (*C*, Fig. 4). The Owl, Acorn and other lodes are in the granite near two or more small roof pendants.

Canadian Shield

The Canadian shield has long been known for its valuable deposits of nickel-copper ore and other minerals, but relatively small deposits of gold ore were mined in the last century, particularly in the gold-arsenopyrite-bearing area west of Kingston, Ont., and in the Lake of the Woods region. In the early part of the present century, however, a remarkable development was begun which resulted in the discoveries of Porcupine, Kirkland Lake and a number of gold-bearing districts of lesser importance. The great production of these areas has stimulated prospecting over the entire Canadian shield and new discoveries are constantly being made.

The country is an area of greenstones, schists, gneisses, etc., intruded by pre-Huronian granite and gneisses probably belonging chiefly to the Algoman period. There are older granites and gneisses within this area but these do not seem to enter into the present problem. All of the important gold deposits are within the areas of the invaded rocks or in the granitic rocks near their contacts with the invaded rocks or in and

around small trough cupolas. Parts of the area of the Canadian shield are covered, but large parts are exposed and have been closely mapped and described in various reports of the Canadian Geological Survey, the Ontario Bureau of Mines and the Manitoba Industrial Development Board. These reports show the general nature of the deposits and their relations to the geological setting and the structural control.

As shown by the maps of Miller, Knight, Burrows, Bruce, Collins, Cooke, DeLury, Quirke and others, there are two great auriferous belts in east Ontario, each corresponding to a great mass of the older rocks surrounded by the granite. The north belt contains Porcupine, Dome, Munro, Lightning and other districts and the south belt contains Mid-

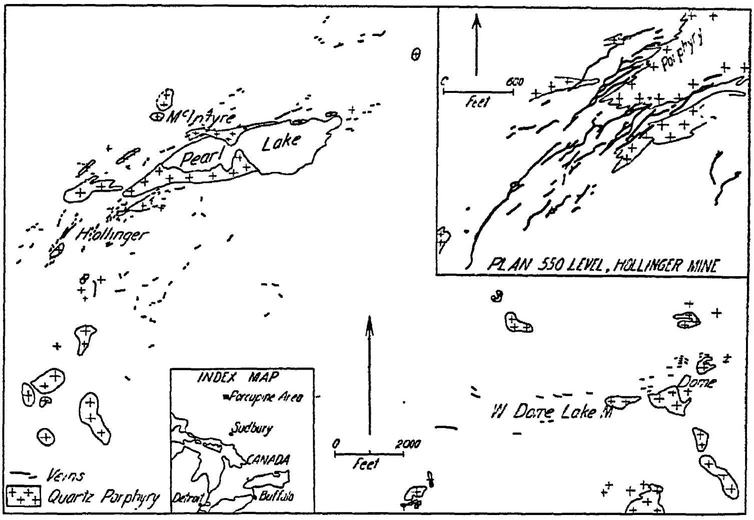

FIG. 5.—PORCUPINE AND DOME GOLD-BEARING DISTRICTS, ONTARIO, WITH PART OF HOLLINGER MINE, 550-FT. LEVEL. (*Data from Burrows, Ontario Dept. of Mines.*)

lothian, Matchewan, Kirkland Lake and Larder Lake. This belt extends into Quebec and on it are developed Rouyn, Stabell and other copper and gold-bearing areas.

The Porcupine[1] district of eastern Ontario (Fig. 5) is in a great eastward-trending "roof pendant" that lies in the Algoman granite. It is the most productive gold-bearing area in North America and one of the most productive in the world. The principal mines are the Hollinger, McIntyre and Dome, and several other important mines are under development.

[1] A. G. Burrows: The Porcupine Gold Area. Ont. Dept. Mines (1924) Pt. 2, 1–112; 33*d Ann. Rept.* see also reports in volumes **20, 21** and **24** of the same department.

E. Y. Dougherty: Mode of Formation of Porcupine Gold Veins. *Econ. Geol.*, (1920) **20**, 660–670.

The Keewatin and Temiskaming schists and related rocks are intruded by many stocks of Algoman granitic porphyry. Veins are found in the schists near the porphyry and subordinately in the porphyries themselves. Although the granite and the porphyries are somewhat schistose, they are much less metamorphosed than the rocks which they intrude. (See Table 1.)

TABLE 1.—*Rocks Exposed Near Porcupine* (*Data from Burrows*)

Keweenawan.—Olivine diabase intrusives.

Matachewan.—Quartz diabase intrusives.

Algoman.—Granite, granite porphyry, quartz porphyry, feldspar porphyry; ore probably connected with the granitic intrusions; much less metamorphosed than Temiskaming sediments; no later granite known near Porcupine.

Haileyburian.—Serpentine.

Temiskaming.—Conglomerate, graywackes, slates, quartzites.

Unconformity.

Keewatin.—Basic and acidic lava flows; basalt, andesite, dacite, rhyolite, tuffs and agglomerates now altered to gray and green schist. Carbonate schist, iron formation, slates. Main veins of Porcupine are in Keewatin.

The chief ore deposits are veins which are relatively short and have great width. They are developed several hundred feet on strike, are generally from 4 to 30 ft. wide and extend downward probably more than 3000 ft. They form complicated systems rudely parallel as shown by the map of the Hollinger mine, Fig. 5, after Burrows. The veins of the Hollinger, McIntyre and neighboring mines seem to be grouped around the intrusive porphyry mass of Pearl Lake. The veins in certain areas are very complexly faulted. The ore carries pyrite, chalcopyrite, sphalerite, galena, a little pyrrhotite, scheelite, graphite and other minerals. Tellurides are rare, but not unknown. The gangue is composed of quartz, carbonates, etc. The wall rock is sericitized and carbonate is generally present; tourmaline is noted. Much of the ore is altered replaced wall rock. The ore carries $7 in gold per ton, or more; very little silver is present. The ore is primary and no zoning is known.

The Dome mine is about 2½ miles southeast of the Hollinger mine. It also is near a group of porphyry intrusions. Veins are found in both the Keewatin and Temiskaming rocks. They are more irregular than in other properties and large low-grade bodies of gold-bearing quartz have been worked in open pits.

The Kirkland Lake[2] gold area is in the Larder Lake mining division, district of Temiskaming, Ontario. Gold was discovered in the area in

[2] E. W. Todd: Kirkland Lake Gold Area: Ont. Dept. Mines, 37*th Ann. Rept.* (1928) Pt. 2, 1–175.

A. G. Burrows and P. E. Hopkins: Kirkland Lake Gold Area. *Idem* (1925) **32**, Pt. 4, and **23** (1914) Pt. 2, 1–32.

E. L. Bruce: The Swastica Gold Area. Ont. Bur. Mines (1912) **21**, Pt. 1, 256–265.

1911. Production began in 1915 and steadily increased until the district became one of the greatest gold-producing districts of Canada. The rocks of the Kirkland Lake area are all of pre-Cambrian age. Volcanic tuffs and other volcanic rocks are overlain by conglomerates and graywacke. These rocks are of Temiskamian age and form part of a great

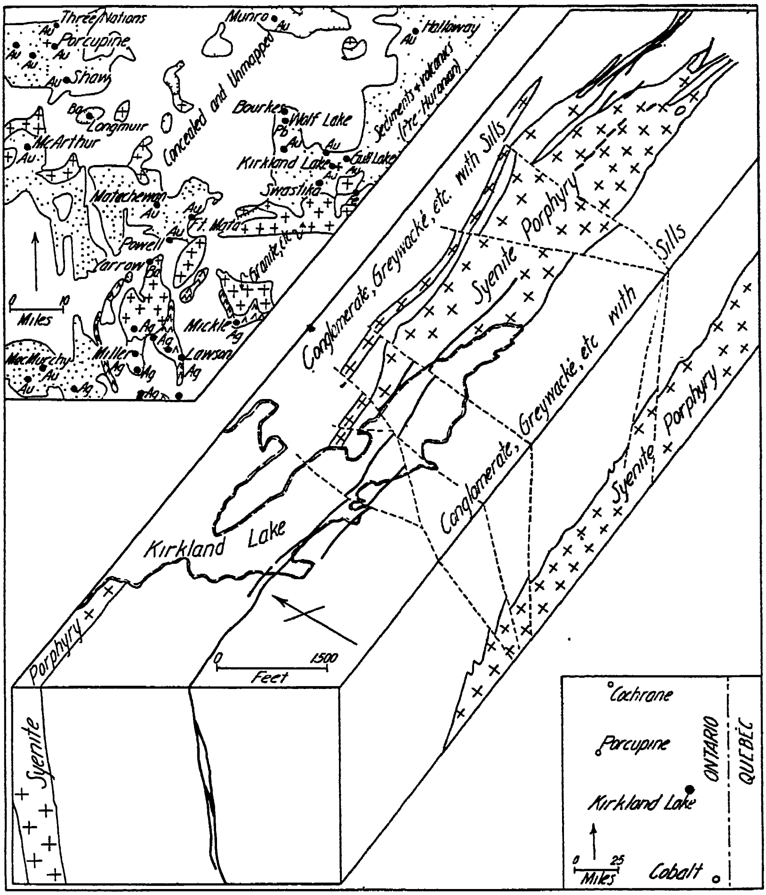

FIG. 6.—GENERAL RELATIONS OF AURIFEROUS DEPOSITS OF KIRKLAND LAKE ONTARIO, TO INVADING AND INVADED ROCKS. (*Based on diagram by E. W. Todd. Ontario Dept. of Mines. Inset after Miller, Knight, Collins, Quirke and others.*)

Many of smaller sills are omitted. Upper inset shows general relations of mining districts of parts of Ontario to intruding and intruded rocks. Broad lines are auriferous veins.

east-west geosyncline that extends eastward from Matachewan 75 miles to the Quebec boundary and 125 miles beyond. These rocks are intruded by (1) basic syenite and lamprophyre, (2) red syenite, (3) syenite porphyry and (4) lamprophyre and diabase dikes and sills all of Algoman age. The basic syenite, red syenite and syenite porphyry are named in

order of age and of increasing silica. They are probably all differentiates from the Algoman granite, as stated by Todd. The ore deposits followed the intrusion and fracturing of the porphyry and are probably connected genetically with the underlying intrusion which supplied the porphyry magma. Basic dikes intrude the rocks named. A large bosslike mass of the porphyry lies east of Kirkland Lake (Fig. 6) and is the central feature of the mineralized area. Many dikes and sills of the porphyry are intruded in all of the older rocks and some of them seem to make off from the central boss. Both the bedded rocks and sills dip steeply. They are much faulted by a series of strike faults and also by a series of later faults nearly normal to the strikes of the rocks.

The ore deposits are great fissure veins in part replacements of the wall rocks. They lie in or against all rocks of the area. They fill faults, some of which, as shown by Todd, have considerable throws. The veins are found chiefly in a fault zone called the "Main Break." The principal faults of this zone follow a line about N. 68° E. and along this line the producing mines are strung out over an area two miles long. The ore-filled faults are faulted by cross faults that strike nearly north, and the veins and cross faults are faulted by nearly flat faults.

The veins carry quartz, carbonate, gold and small amounts of sulfides including pyrite, chalcopyrite, sphalerite and galena. In some of the ore there is considerable tellurium although very little of the gold occurs as tellurides. The sulfides make up less than 2 per cent of the ore and there are no large masses of sulfides. The syenite and syenite porphyry were more readily shattered along the faults than the tougher older rocks and offered more open ground to the mineralizing solutions. There is strong hydrothermal alteration of the walls, consisting of sericitization and carbonation. Much of the ore is the altered country rock containing quartz and gold. There is no zonal arrangement in the district and little or no secondary enrichment.

The western part of Quebec adjoins the gold-bearing area of Ontario and has similar geological features (Fig. 7). In the Rouyn district the rocks are pre-Cambrian and belong mainly to the pre-Huronian group, which includes schists, basalts, andesites, dacites, rhyolites, tuffs, sediments, etc. These rocks are closely folded and commonly strike east. They were intruded by amphibolite and diorite porphyry before folding, and by "older gabbro," quartz diorite, granite, syenite porphyry and other rocks after folding. The older gabbro and granitic rocks are rarely schistose. The Cobalt sediments, almost flat-lying, rest unconformably above the earlier series including the granitic intrusives.

The ore deposits are lodes replacing various rocks in shattered zones and lie in rocks older than the Cobalt series. The region of ore-bearing rocks is almost surrounded by granites. Many of the deposits are near syenite porphyry. There is probably a genetic connection between the

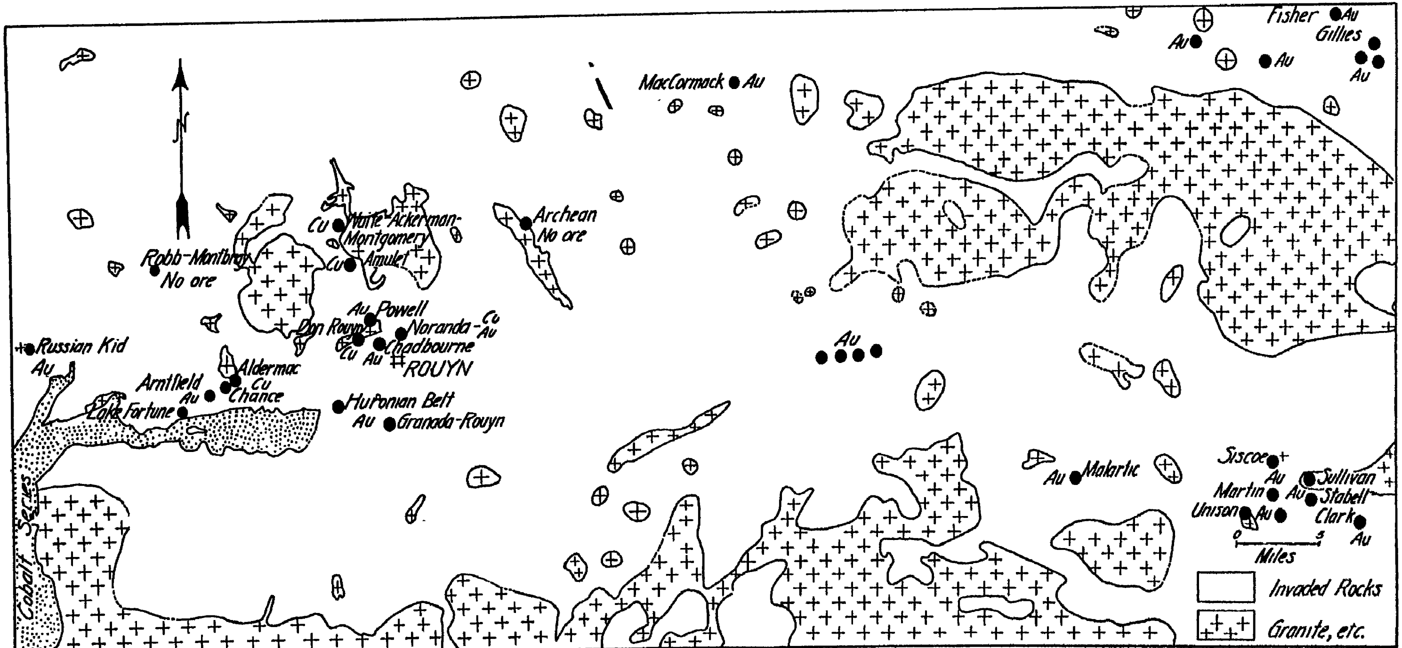

FIG. 7.—ROUYN DISTRICT, WESTERN QUEBEC. (*After Cooke, James, Mawdsley, Harvie, Tanton, Walker and others, of Geol. Survey of Canada.*)

Shows relations of deposits of copper and gold to intruding granite and porphyry. Nearly all the deposits are in invaded rocks near the invading granites and porphyries. A few small deposits are in invading rocks.

deposits and the granitic rocks and porphyry.[3] The small porphyry stock may represent places where the underlying magma rose to relatively high places and the copper and gold deposits were formed probably in the main by solutions rising from cupolas in the great roof pendant. That hypothesis, as already stated, is supported by relations of deposits to dikes and stocks in many other deeply truncated intruded areas.

The deposits are in the invaded series, generally not far from the granite contacts or near the syenite porphyry intrusions. There are large areas of granites but no important deposits are found in them. The area is faulted and some of the faults are later than the ores.

The copper ores, as stated by Cooke, consist chiefly of replacements in the ancient volcanic series. Many of them are almost pure sulfide ores. The minerals include pyrite, pyrrhotite, sphalerite, chalcopyrite and magnetite. The deposits were formed at relatively high temperatures. They are short broad lenses in shear zones and in general the chalcopyrite ore is found around ends of and enveloping the pyritic bodies and filling cracks in them, showing that it is of an episode later than the deposition of pyritic ore. The rocks replaced are rhyolites, dacites and their breccias and tuffs. Many of the deposits, including some of the lenses of the Amulet mine, are found in folds, and these in general lie near the crests of anticlines and are in fragmental rocks at tops of flows that are covered by relatively impermeable rocks. The age of the ores is about the same as the age of the syenite porphyries. In the Aldermac mine the pyritic ore was formed before and the chalcopyrite ore after certain porphyry dikes. Rocks near the ore are highly silicified and chloritized.

The Horne (Noranda) mine, which has opened the largest deposits in the Rouyn district, is about one mile north of Rouyn in an area of greenstones, dacites, rhyolites, rhyolite breccias and tuffs which are intruded by quartz diorite, by syenite porphyry dikes and by the "later" gabbro. The mine lies about one mile east of a small granitic intrusive. The ores are in the volcanic rocks and tuffs which are greatly altered and are found subordinately in the quartz diorite ("older gabbro"). The volcanic beds are highly folded and locally are on edge. The rocks have been sheared and shattered and the ores are localized in part by the zones of drag folding. Dense lavas and porphyries were not replaced, but served

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[3] M. W. Wilson: Timiskaming County, Quebec: Can. Geol. Survey *Mem.* 103 (1918) 1–197.

W. J. Wilson: Geological Reconnaissance Along the Line of the National Transcontinental Railway in Western Quebec. Can. Geol. Survey *Mem.* 4 (1910) 1–52; also *Mem.* 17 and *Mem.* 34.

H. C. Cooke: Geology and Ore Deposits of the Rouyn-Harricanaw Region, Quebec. Can. Geol. Survey *Mem.* 166 (1931) 1–314.

H. C. Cooke: A Correlation of pre-Cambrian Formations of Northern Ontario and Quebec. *Jnl. Geol.* (1920) **28**, 304–332.

as dams which controlled the courses of solutions. The largest deposit is the H orebody, which on level 1 is 150 ft. wide, strikes N.60° E. and dips 80° SE. On level 1 the lode carries $1.54 gold and 0.5 per cent copper; on level 3 it is richer and at a depth of 975 ft. it is high-grade copper-gold ore. In the Horne, or Noranda, group there are 10 lenses of ore estimated to contain 3,426,000 tons with 7.53 per cent copper and $3.29 gold per ton; also 3,000,000 tons of lower grade ore with 2 per cent copper and $3 gold per ton.

The Amulet copper mine is five miles north of Noranda. Six irregular bodies of copper ore are found in the lavas and in rhyolite, near intruding granite. Most of the deposits lie in or near the crests of anticlines in the porous amygdaloidal or fragmental tops of the rhyolite flow, which are covered over by relatively massive and relatively impervious dacite. The deposits are flat pancakelike masses that conform in a general way with the contacts. The ore contains pyrite, pyrrhotite, sphalerite and chalcopyrite. There were developed in 1929 about 400,000 tons of ore with 2.78 per cent copper, 12.48 zinc, $1.04 gold and 2.9 oz. silver per ton. Since then additional reserves have been developed. In the W.A.M. (Waite-Ackerman-Montgomery) mine north of Amulet a lens of sulfide is enclosed in lavas and probably is located on an anticline. The main deposit has a central core of chalcopyrite ore surrounded by a deposit high in zinc. About 428,310 tons of zinc and copper ore are developed.

At the Don Rouyn mine, three miles west of Noranda, a narrow shattered zone in granite is mineralized with chalcopyrite. At the Robb Montbray mine, 11 miles northwest of Rouyn, some high-grade gold-copper ore is found in chloritic schist.

The Aldermac mine is 10 miles west of Noranda near the south end of a small intrusive of syenite porphyry, a finger of which points to the mineralized area. The ore is in dark lava and consists of pyrite-pyrrhotite ore with chalcopyrite. The pyrite-pyrrhotite ore is earlier than dikes of porphyry which cut the lavas but the chalcopyrite ore, which partly replaced the pyritic ore, is found also in the porphyry dikes.

The deposits described above are primarily copper deposits, but the Horne and Robb-Montbray ores carry important amounts of gold and smaller amounts are present in other deposits. The Chadbourne and Powell mines, near Rouyn, have opened gold-bearing deposits. Southwest of Rouyn, gold ores are found in Lake Fortune, Arntfield and other lodes, and some 50 miles east of Rouyn gold lodes are found at many places. The Siscoe mine, on Siscoe Island in Lake de Montigny, opened a network of small auriferous veins in a small granodiorite intrusive. Three miles southeast of Siscoe mine, in the Sullivan mine, gold veins with quartz and tourmaline and some sulfides are found. At the Stabell mine, one mile south of the Sullivan mine, veins are found in volcanic rocks. These carry chalcopyrite very high in gold.

The Eastern Ontario gold belt (Fig. 8) is about 40 miles west of Kingston. Gold was discovered in the area[4] in 1866 and the mines were in successful operation many years before Cobalt and Porcupine were discovered. The country is an area of pre-Cambrian rocks including limestone, talc schist and conglomerate, which are intruded by diorite and still later by granite. Paleozoic limestone rests unconformably upon these rocks in the south part of the area. The chief deposits are lodes

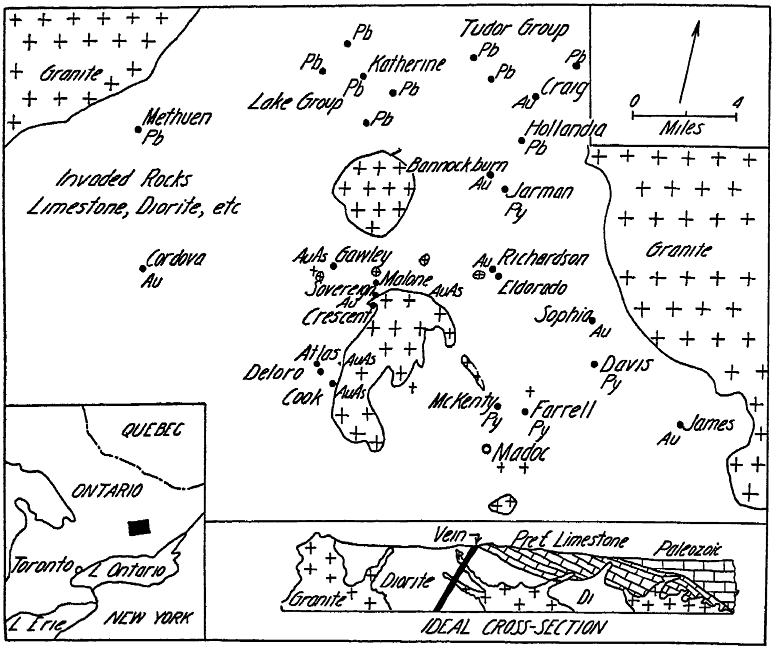

FIG. 8.—EASTERN ONTARIO GOLD BELT. *(Map and cross-section from data and maps of W. G. Miller; lead mines and some others added from maps by Uglow, Adams, Barlow and others.)*

of arsenopyrite and gold, which, as stated by Miller, were deposited by solutions derived from the intruding granite. Miller notes that galena is found in the upper part of the Cordova gold mine and lead ores are found at several places to the north of the gold-arsenic mines. The arsenopyrite-gold lodes are near the central granite mass (Fig. 8) whereas pyrite deposits and gold deposits without arsenic are at some distance from the mass. This rude zoning of the ores seems to have been recognized by Miller 30 years ago. It is noteworthy that this is the only gold-bearing area in any of the four ancient shields that shows even a rudimentary zoning.

[4] W. G. Miller: The Eastern Ontario Gold Belt. Ont. Bur. Mines, 11*th Ann. Rept.* (1902) 186–207.

Siberia

In the north central part of Siberia[5] is an area of pre-Cambrian rocks, strongly folded, that are covered at places by sediments of later age that are more gently deformed. This great area known as the Siberian shield has not been closely folded since pre-Cambrian time. It is nearly surrounded by a broad belt of rocks that were folded in early Paleozoic time. Still further east and south outside of the belt of early Paleozoic folds are folds of the later, Hercynian, age.

Great masses of intruding granitic rocks are found over large parts of south central and southeastern Siberia and with these are associated scores of gold fields, many of which have yielded large amounts of placer gold. Most of these fields are not strictly within the pre-Cambrian shield area. The granite masses are not so deeply eroded as they are in Canada, Southern Rhodesia and Western Australia. Gold in roof pendants is found only rarely. The gold-bearing areas are framed with the invaded rocks rather than the intruding granites. The metallization is embatholithic (IV) rather than endobatholithic (V). It is of the Sierra Nevada and Coast Range batholith type rather than the more deeply eroded type of Canada, Southern Rhodesia (V) and Western Australia (V).

The gold fields of Siberia are among the most productive in the world. Since most of the gold has been derived from placers, there are obvious difficulties in the interpretation of the metallogeny of the region, yet the maps issued by the Russian Geological Surveys are sufficiently detailed to permit very probable inferences as to the sources of the gold. Parts only of the auriferous areas are strictly within the areas of the pre-Cambrian shield. As stated, many are in the broad marginal region of early Paleozoic folding and of pre-Middle Devonian granitic intrusions. The gold is obviously of local origin. The relation of the placers to the invading granites is obvious from inspection of the maps of the principal fields. In the Yenissei district that lies north of Yenisseisk, in the angle between the Yenissei and the Angora Rivers, the gold is found chiefly

[5] The geology of parts of the gold-bearing area of Siberia was mapped in reconnaissance by W. Obroutchew, A. Guerassimow and Prince Guedroic. A résumé of their papers and reproductions of certain maps were issued by L. DeLaunay [*Ann. des Mines*, ser. 10 (1904) **15**, 350–359; also in 1911 in a volume entitled La Geologie et Les Richesses Minérais de l'Asie, 524.] The Bodiabo report by Obroutschew, in Russian, appeared in the series entitled Explorations Géologiques dans les Régions Auriferes de la Siberia (1901 and 1903) **1** and **2**. Maps by Guerissimow, Preobrajensky and D. Mouchketow appear in volume 5 of the same series and one by V. Kotoulsky in volume 10. Most of the older Russian papers are abstracted in French. Several abstracts in German have been published, one by J. Ahlberg entitled Die neueren Fortschritte in der Erforschung der Gold-Lagerstätten Siberiens [*Ztsch. f. prakt. Geol.* (1913) **21**, 116]. Since 1919 many short bulletins in Russian have appeared.

outside of the areas of the invading granites, in the metamorphic rocks intruded by the granites, and gold has been mined from conglomerates that lie at the base of the Cambro-Silurian rocks. Since the gold deposits obviously are related to the granitic intrusions, they are either of pre-Cambrian or very early Paleozoic age.

The Bodiabo (Olekma) gold field (Fig. 9) in the upper Lena basin between the Lena and Vitim Rivers. The chief town is Bodiabo on the Vitim, from which a short railway is built into the main gold field. The rocks of this region are schists, slates, quartzites and limestones closely folded, commonly overturned and intruded and metamorphosed by granite. Quartz veins with pyrite and some gold are numerous and have contributed gold to the placers. They are so poor that generally they are not worked, but it is not improbable that they represent the low-grade roots of veins that were richer in the upper parts, which now are eroded. The rocks are pre-Cambrian or early Paleozoic, or both, and the gold lodes are believed to be of early Paleozoic age. The chief placer deposits are in a belt of slates and schists 60 km. wide. A great granite mass 16 miles wide or more crops out northwest of the gold field and another large mass is found along the Vitim River to the south of the gold field (Fig. 9).

The Bodiabo district contains some of the richest placer deposits of Siberia. Almost all of them are perpetually frozen and contain gravels of granite, schist, quartz and some of rocks foreign to the region that are said to bear glacial striae. The valuable ore commonly lies from 65 to more than 200 ft. deep and the gold is won by underground mining. In a hole reported by Purington,[6] the supposed glacial material lies 140 ft. deep. It has been stated that the gold has been transported by the glaciers, but it does not appear probable that it has been transported far, since the gold-bearing area seems to be confined to the area between the granite masses. Moreover, the gold found at great depth is rough and coarse, and some of it appears in crystals. In the auriferous stratum schist predominates and much pyrite, doubtless of local origin, is present. The deposits are said to belong to the "valley placer" type. In this region the Cambrian as well as the pre-Cambrian rocks are folded and Silurian rocks lie unconformably above them. It appears probable that the granites were formed and the gold lodes deposited in Cambrian or pre-Cambrian time.

In the upper Yenissei drainage basin, valuable gold deposits are found south of Marinsk and west of Minusinsk in the region between the Tom and Abakan Rivers. In this region schists, quartzites, volcanic rocks, etc., are invaded by granites, and considerable parts of the area are covered by Middle Devonian and later sedimentary rocks, and by

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[6] C. W. Purington: *Min. Jnl.* (1907) **81**, 517–518.

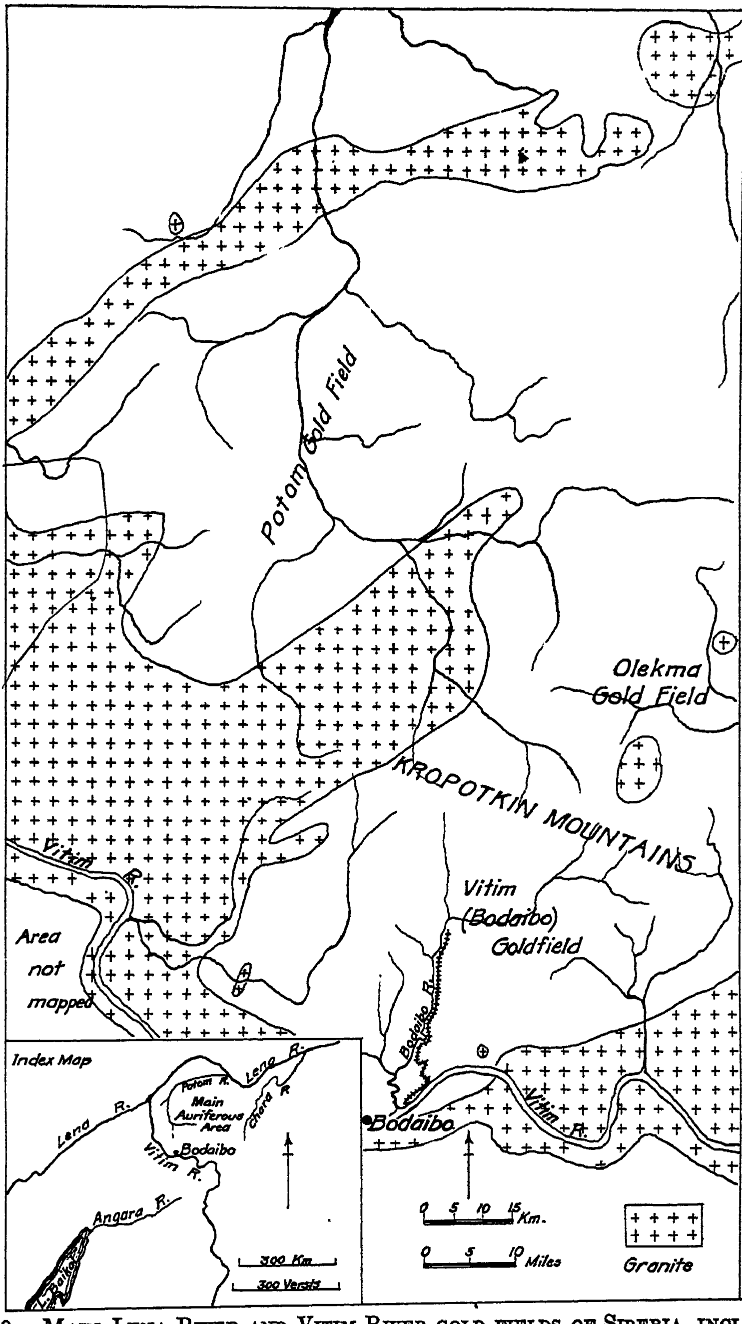

FIG. 9.—MAIN LENA RIVER AND VITIM RIVER GOLD FIELDS OF SIBERIA, INCLUDING BODIABO AND OLEKMA. (*From Guerissimow, Obrutschew and others.*)

Granites are intruded in slates and schists. The chief gold deposits are placers and found in areas of invaded rocks near granites.

basic lavas. These rocks are later than the granite and also later than the gold deposits. Gold lodes have been worked in several mines. These lie in the invaded rocks and in the granites near the invaded rocks (Fig. 10).

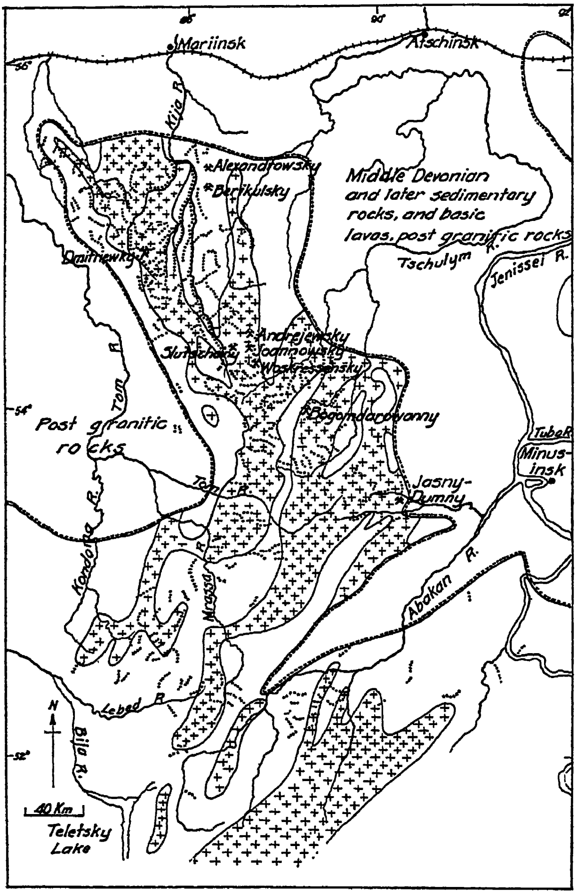

FIG. 10.—YENISSEI RIVER GOLD FIELDS IN REGION BETWEEN TOM AND ABAKAN RIVERS. (*After Bogdanowitch, Rentowsky and Obrutschew.*)

Areas shown white are intruded by granitic rocks (indicated by crosses) except those outside of the double lines where Middle Devonian and later sediments and lavas cover older rocks unconformably. These are later than granite and no gold lodes have been discovered in them. A few gold lodes have been worked near margins of granite. Lines of dots show auriferous gravels.

The quartz mines are small compared to the placers, although the Bogom-darowanny mine for a number of years produced more than $100,000 annually. The map gives the impression that these lodes are farther within the granite borders than are similar gold deposits in such positions in other parts of the world, but the descriptions of the lode mines

show that most of them are in contact phases of the granite and that marginal basic rocks are included in the granite as mapped on Figs. 10 and 11. The placers are mainly in the narrower intrusions and in the invaded rocks. A considerable part of the area shown in Fig. 10 is covered with post-granitic rocks in which placers are practically wanting.

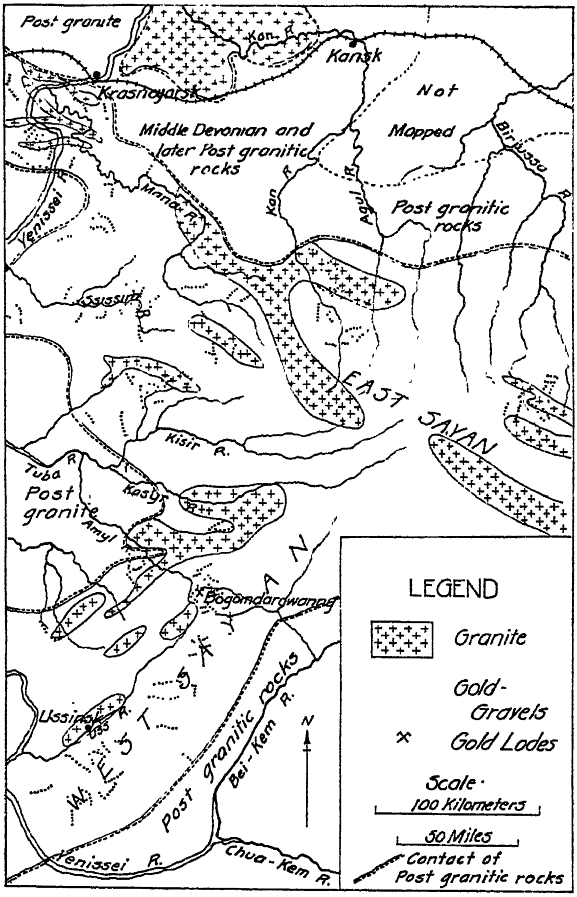

FIG. 11.—SAYAN MOUNTAINS REGION JUST EAST OF MINNUSINSK AREA, JOINING ON EAST SIDE THAT SHOWN ON FIG. 10. (*From Bogdanowitch, Rentowsky and Obrutschew.*)

Areas shown white are intruded by granite except those enclosed with double lines, which show areas of post-granitic rocks. Gold placers are found chiefly in areas of invaded rocks. Lines of dots show auriferous gravels.

In the area south and southeast of Krasnojarsk, in the East Sayan and West Sayan Mountains (Fig. 11, which joins Fig. 10 on the east) the granitic intrusions are less closely spaced. The placers are essentially confined to areas above the invaded rocks and are more sparingly developed above granite. In this region also there are large areas of post-granitic barren rocks (Fig. 11).

India

The Kolar[7] district, Mysore, India, which is one of the world's greatest gold fields, is also in an ancient shield. It is in a belt of ancient schists consisting of hornblende schists, in part altered volcanics, of quartzites and jaspers. This belt (Fig. 12) is 50 miles long and 4 miles wide and is nearly surrounded by gneisses and granites. There is an older granite gneiss, an older porphyritic and gneissic granite and a "newer granite" which intrudes the schists. The relations of the older granite and gneiss to the belt of schist are not determined. The gold deposits lie in schist at the south end of and are aligned with a mass of "newer" granite, known as the Peninsular granite, which occupies great areas in southern India. The gold of the area has been derived almost exclusively from the Champion lode which is opened in the Mysore, Champion, Ooregum, Nundydroog and Balaghat mines. The productive part of the lode is about 4½ miles long and the deepest levels are about 6000 ft. below the surface. The lode has produced more than $300,000,000 gold, about one-third of which was profit.

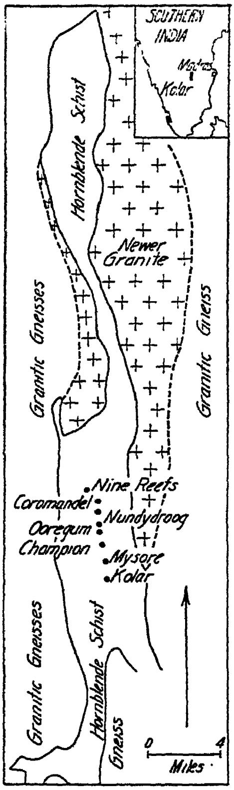

FIG. 12.—POSITIONS OF CHIEF GOLD DEPOSITS IN KOLAR GOLD FIELD, MYSORE, SOUTHERN INDIA. (*After a geologic map by W. F. Smeeth. Mine map by F. H. Hatch.*)]

The Champion lode strikes NNW., dips with the schists about 55° west but steepens with depth. The main oreshoots plunge about 45° north, so that with the dip and the rake they are comparatively flat lying. Between the oreshoots the lode commonly pinches to a narrow width or is barren. At places it is cut by faults. In certain areas the lode is doubled over like the letter S. It is commonly supposed that the lode has been folded although this theory is discarded in a recent report by Pryor, who regards the so-called folds as fractures. If the lode was not folded it appears possible that it was formed in part by replacement

[7] F. H. Hatch: The Kolar Gold Field. Geol. Survey India *Mem.* 33 (1892) 1–81.

B. Foote: Report on the Auriferous Tracts in Mysore. Geol. Survey India *Record* (1888) **21,** 40.

T. A. Rickard: Persistence of Ore in Depth. *Trans.* Inst. Min. and Met. (1915) **24,** 3–46.

W. F. Smeeth: Records of Mysore Government Press.

T. Pryor: The Underground Geology of the Kolar Gold Field. *Trans.* Inst. Min. and Met., (1924) **33,** 95–135.

of a favorable bed which before metallization had been folded into the carinate folds the ore now occupies. Many of the oreshoots follow the folds plunging flatly northwest. Their descriptions recall the structure of the deposits of Homestake mine, South Dakota, and the copper lodes of Ducktown, Tenn. The ore carries quartz, biotite, pyrite, pyrrhotite, arsenopyrite, chalcopyrite and galena. Actinolite, tourmaline and pyroxenes and some calcite are closely associated with the mineralization. The veins were deposited chiefly along planes of foliation of the schists. Unaltered dolerite dikes traverse all of the rocks named above and also cut the lodes, and pegmatitic and aplitic granites are found in the mines along the lode.

Pryor thinks that the mineralization is due to the older gneissic granites. From this, however, Hatch, Holland, Campbell and others dissent. As stated by Campbell, the Peninsular gneiss or "newer granite" is the general country rock of southern India whereas the schists and older gneisses occupy comparatively small areas. As stated by Holland,[8] the whole country is "immersed in a bath of Peninsular granite." If the lodes are related to the "younger" granite they lie near a point of the parent intrusive. This recalls the relations of the lodes at Kalgoorlie. The main gold-bearing belt may be situated above a high point in the underlying igneous mass.

Southern Rhodesia

In Southern Rhodesia[9] the oldest rocks are in the Swaziland system, which consists of a lower division of greenstones that include basic and other lavas, associated sedimentary beds and intruding diabase dikes and sills, and above these unconformably a series of arkoses, graywackes and conglomerates (Figs. 13 and 14). The Swaziland group is intruded by great masses of granites, gneisses and granitic porphyry, which together cover the larger part of the mineral-bearing region of Southern Rhodesia. The intrusions of earlier granites and associated rocks were followed by the deposition of the Frontier quartzites and talc schists and after that the Lomagundi sedimentary rocks and andesites. Following the deposition of the Lomagundi beds, there was a series of younger granitic

[8] Holland: *Trans.* Inst. Min. and Met. (1924) **33.**

[9] H. B. Maufe: Map of Southern Rhodesia issued by the Southern Rhodesia Geological Survey; also an outline of the geology of Southern Rhodesia, Short Report 17 (1924).

H. B. Maufe and others: Southern Rhodesia. Int. Geol. Cong. Session 15. Guide Book C20 (1929) 1–64.

E. H. Garthwaite: *Min. Mag.* (1906) **13,** 1–10.

E. P. Mennell: *Idem* (1913) **9,** 203–205.

T. F. Van Wagenen: Four Typical Rhodesian Gold Mines. *Min. & Sci. Pr.* (1900) **91,** 313–314.

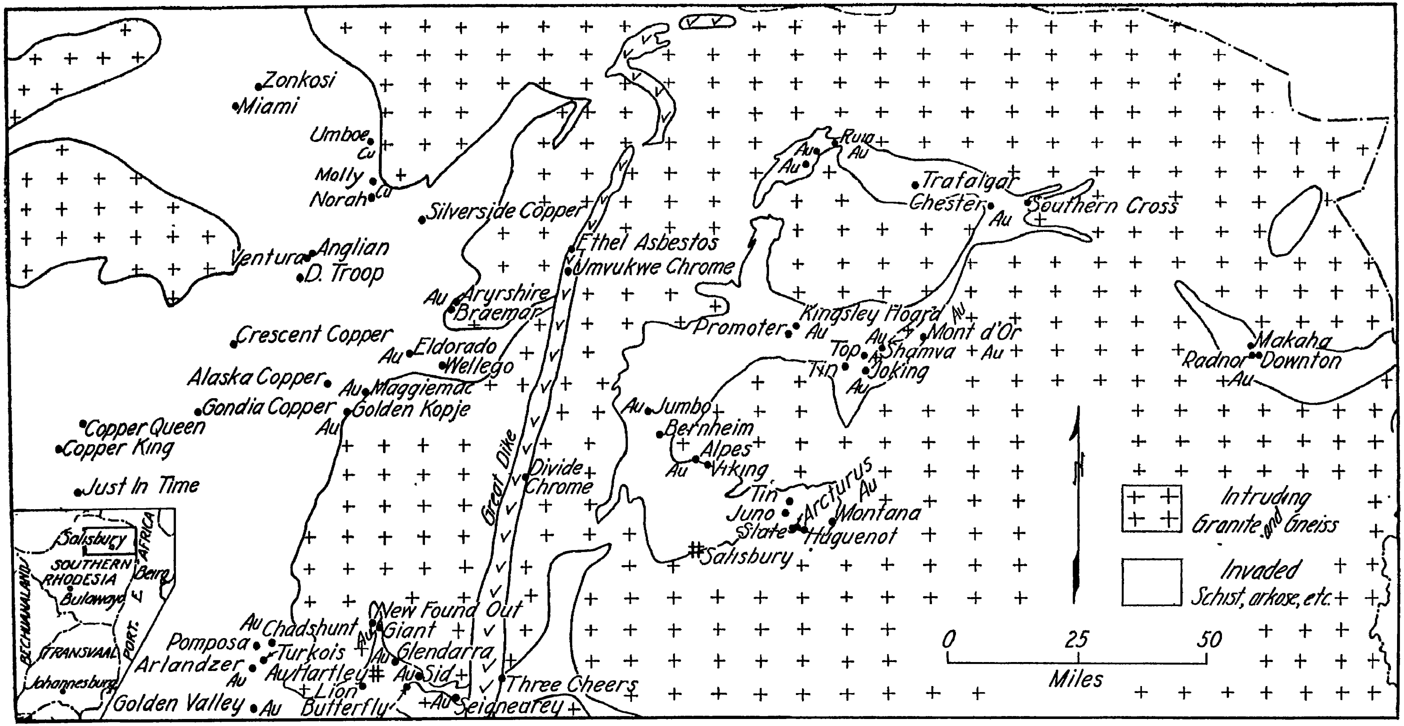

FIG. 13.—RELATIONS OF GOLD DEPOSITS TO INVADING GRANITES, SOUTHERN RHODESIA AURIFEROUS REGION. (*Data from Maufe and others of Southern Rhodesia Geol. Survey.*)

Gold lodes are in and near islands of invaded rocks and were deposited by solutions from invading rocks.

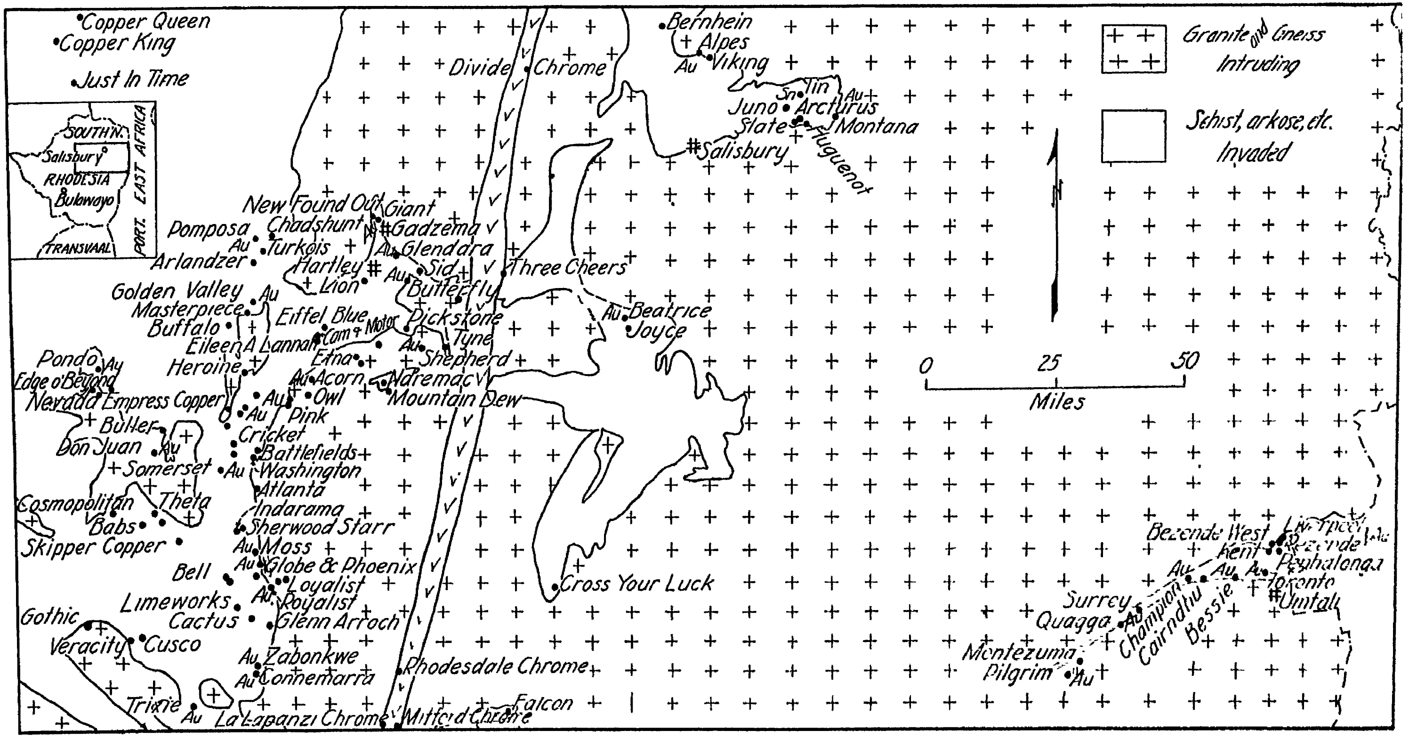

FIG. 14.—RELATIONS OF GOLD DEPOSITS TO INVADING GRANITES, SOUTHERN RHODESIA AURIFEROUS REGION. (*Data from Maufe and others of Southern Rhodesia Geol. Survey.*)

Gold lodes are in and near islands of invaded rocks and were deposited by solutions from invading rocks.

intrusions and the Great Dike was formed, the latter being a basic body 325 miles long and some 4 to 6 miles wide. All of these rocks are pre-Cambrian. At places later rocks rest upon them but these are of little importance to the present discussion. Great islands of rocks of the Swaziland system are surrounded by the granites and at many places are intruded by small dikes and stocks of granite or by granitic porphyries. Gold lodes are closely spaced. As in Western Australia, there are hundreds of mines and groups of mines, but there is no outstanding district like Kalgoorlie, although there are a score of important though smaller districts. All of the large deposits are in the invaded rocks or in the intruding granites near the contacts of the granites with older rocks. A few small mines are found in granite more than a mile from the contacts (Figs. 13 and 14).

The Shamva, which has been one of the most productive mines, is in quartzite at the end of a small mass of intruding granite. The Wanderer, Cam and Motor, Globe and Phoenix, Gaika, Lonely, Grant, Eldorado, Aryshire, Washington and many other deposits are in the invaded series. In the Invincible group near Battlefields the lodes that have had a moderate production are in granite between the main mass of the invaded rocks and a small body of the invaded rocks which is surrounded by granite. The Owl mine also is in granite within one-half mile of two small bodies of the invaded series. The Acorn and several other small mines in granite are even farther from the contact. The Unice, a very small mine east of Battlefields, which together with the Zurich near by produced 2846 oz. gold, is in granite one and three-quarters mile from the contact. This deposit is one of the few gold lodes that lie in granite at some distance from the contact and is a representative of a very small and relatively unimportant group.

A comprehensive bulletin by A. M. Macgregor,[10] covering the area between Galooma and Battlefields and extending somewhat beyond these camps, has appeared recently. This large area lies between the Cam and Motor mine on the north, which is now Southern Rhodesia's most productive gold mine, and the Globe and Phoenix and Gaika mines on the south. It contains 120 small mines but no large ones. Macgregor notes that within this area the Swaziland "schists" at places are not much metamorphosed by pressure but that lavas still contain their vesicles with original outlines and that the gneissic texture of the granite is not due to pressure but is original. In this ancient area dynamic metamorphism has not gone so far that relations are obscured. The positions of the gold deposits are in harmony with their positions in other "shield" areas of the world, although a few of the small gold lodes are in granite more than a mile from the contacts. In summarizing the rela-

[10] A. M. Macgregor: The Geology of the Country between Galooma and Battlefields. Southern Rhodesia *Bull.* 17 (1930) 1–144.

tions of the lodes to the granite, Macgregor says: "It is a popular fallacy which used to be accepted almost as commonplace that gold is not found in granite country. In this area granite has been the most prolific producer. A little gold, moreover, has been won from the quartz porphyry stocks. Together these allied rocks have yielded 69 per cent of the gold won in the area."

It is noteworthy that the lodes in the granite, including the Battlefields group, the Owl, Acorn and others of the larger mines, are all very near the contact on the edges of the granite bodies or very near one or

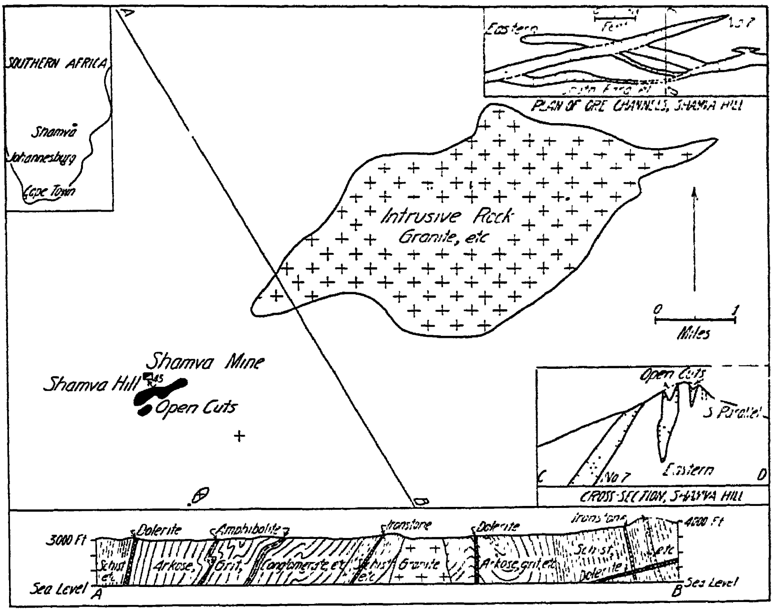

FIG. 15.—SHOWS POSITION OF SHAMVA MINE, SOUTHERN RHODESIA, WITH RESPECT TO PARENT INTRUSIVE. (*Map from report by Tyndale-Biscoe. Insets after Maclaren.*) Mine is in quartzite and lode dips northwest. If intrusive continues its general strike to a point below Shamva deposit, lode with depth would approach more closely ridge of intrusive.

more bodies of schist that are included in the granite. In this area, as elsewhere in Southern Rhodesia and in other shield areas, the granite is essentially barren more than a mile from its contacts with the rocks it intrudes.

The Shamva mine is about 50 miles northeast of Salisbury, in a belt of pre-Cambrian quartzites and other rocks that is about 12 miles wide and intruded by pre-Cambrian granite on either side (Fig. 15). The quartzite body is intruded also by a stocklike mass of granite[11] about five

[11] R. Tyndale-Biscoe: The Geology of the Country around Shamva Mazoe District Southern Rhodesia Geol. Survey *Bull.* 18 (1931).

J. M. Maclaren: The Shamva Gold Mine. *Min. Mag.* (1924) **31,** 329–339.

miles long and three miles wide. The Shamva mine is in quartzite about one mile southwest of the southwest point of the granite stock. The Shamva mine for many years was the leading gold-producing mine in Southern Rhodesia and from 1913 to 1930, when it was closed, had produced about $32,000,000 gold, which is about 90 per cent of the gold produced in the Shamva area treated by Tyndale-Biscoe in the paper mentioned in footnote. The Shamva ore zones are 100 to 200 ft. wide, are enclosed in quartzites and occupied three belts of fractured rock. The ore is commonly associated with much calcite, and calcite is regarded as a favorable indication, which suggests that the beds replaced may have been somewhat calcareous members of the quartzite series, but this is uncertain because there are said to be no reliable markers of bedding in the quartzite series, which is closely folded. The ore carried only about $3 gold per ton. It is said to be as rich in depth as in the portion stoped but that the mine was closed because of the greater expense of the deeper mining.

Western Australia

In Western Australia there is a great area of pre-Cambrian rocks consisting of greenstone, schists, etc., which are intruded by pre-Cambrian granite and gneiss, the latter forming the frame of the picture (Fig. 16) and the invaded rocks forming islands that are surrounded by the intruding granites and gneisses. The gold deposits are nearly all in the invaded rocks, although a few are in the invading rocks within a mile of the contact. The area is deeply eroded and peneplained and in many respects the setting is geologically similar to that of the ancient shields of Canada, Siberia, Southern India and Southern Rhodesia. The deposits are of the deep-seated type and gold is the chief metal.

Kalgoorlie[12] is in the southern part of Western Australia about 370 miles east of Perth. It is the most productive district in Western Australia, having yielded about $400,000,000 gold, which is approximately one-half the gold production of the state. Kalgoorlie lies within a great irregular body of greenstones, schists and related rocks of pre-Cambrian

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[12] F. L. Stillwell: Geology and Ore Deposits of the Great Boulder, Kalgoorlie: W. Aust. Geol. Survey *Bull.* 94 (1929) 1–110 (with atlas).

C. O. G. Larcombe: The Geology of Kalgoorlie. *Trans.* Aust. Inst. Min. Eng. (1912) **14**, 1–327; *Idem* [N.S.] (1927) No. 67, 247–268; also *Min. & Sci. Pr.* (Aug. 12, 1915) 237–245.

E. S. Simpson and C. G. Gibson: W. Aust. Geol. Survey *Bull.* 42.

M. Maclaren and J. A. Thomson: Geology of Kalgoorlie Goldfield. *Min. & Sci. Pr.* (1913) 45, 95, 187, 228, 374.

F. R. Feldtmann: W. Aust. Geol. Survey *Bull.* 69 (1916).

C. S. Honman: The Geology of the Country South of Kalgoorlie. W. Aust. Geol. Survey *Bull.* 66 (1916).

age 100 miles wide and somewhat longer, which forms an island surrounded by granite and gneiss that intrude the schists. The invaded rocks are intruded also by stocks and dikes of porphyry and granite, which in part probably are of the same age as the granite that surrounds the greenstone schist areas.

At Kalgoorlie a great belt of the greenstone series strikes northwest. Intruding porphyry lies on the southwest side of the belt (Fig. 17) and at the southeast end another large body of porphyry intrudes the schist.

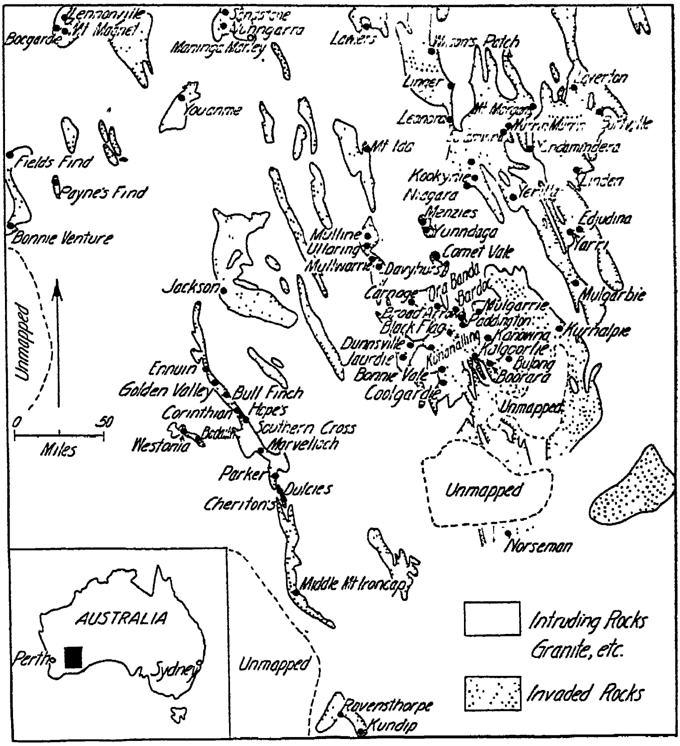

FIG. 16.—PART OF WESTERN AUSTRALIA AURIFEROUS REGION, INCLUDING KALGOORLIE EAST OF CENTER OF MAP. (*From Maitland and others of Western Australia Geological Survey.*)

Gold lodes are in islands in invaded rocks and near islands. They were deposited by solutions from invading rocks.

A finger of the porphyry about two miles long extends northwest from this body and forms a great dike about 300 ft. wide, and similar dikes are found on strike to the northwest. The gold deposits are great lodes, in general nearly parallel to the dikes and to the bedding and schistosity of the country rocks. The most productive lodes lie in the greenstones

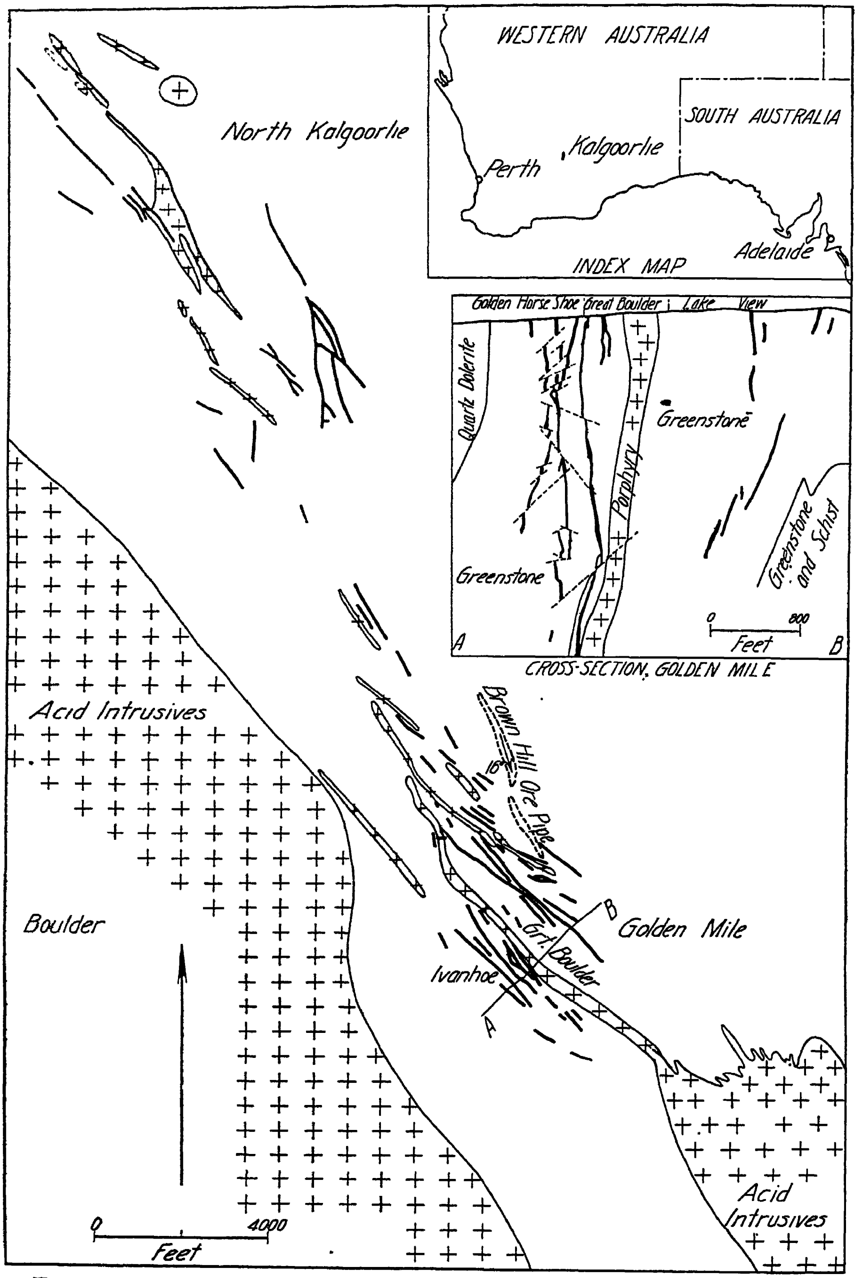

FIG. 17.—KALGOORLIE DISTRICT, WESTERN AUSTRALIA. (*Geology from Gibson, Stillwell, Feldtmann and Finucane. Veins of Golden Mile from Larcombe. Veins of North Kalgoorlie from Feldtmann. Cross-section from Stillwell, Feldtmann and Finucane.*)

Border of intrusive mass of southwestern part of area is largely concealed and contact generalized. Broad lines are auriferous veins.

and schists near the dikes and are in line with the acidic intrusive to the southeast of the district.

The Golden Mile includes the chief deposits, among them the Great Boulder and associated deposits, and the Brownhill-Oroya group. The deposits are in and along zones of fracturing, shearing, sheeting and faulting. In general they strike northwest nearly parallel to the strike of the country and they lie near the belt of intrusive dikes. A noteworthy deposit is the great pipelike body of ore that crops out in the Brownhill claims and descends on a dip of about 16° to the south into the Oroya claim. This mass is formed where one or more faults cross beds that are favorable for replacement; the ore extends out from the intersections along the beds and also along the faults. It is about one mile long and has produced over $35,000,000 gold.

The wall rock along the Kalgoorlie lodes is altered by the deposition of quartz, sericite, chlorite and carbonates in large quantities, with pyrite and gold. At places the gangue includes albite, roscoelite and fluorite. Other minerals are chalcopyrite, sphalerite, galena, pyrargyrite, enargite, löllingite and magnetite. Gold is found both as native metal and in a great number of gold-silver tellurides, including calaverite, petzite, kalgoorlite and others.

The country is peneplained. The waters contain chlorides. Oxidation is attended by breaking down of tellurides. The gold is set free and has accumulated in the oxidized zone. In the presence of such large amounts of carbonate in the ore,[13] gold could not readily migrate downward in the solutions. Some alluvial deposits have formed, but they are relatively unimportant.

Summary

Granitic batholiths are great irregular intrusive masses that slope outward and have undulating roofs. Unlike laccoliths, they are not known to have floors. Most lode ores of the metals are formed in the roofs and near the roofs of such batholiths by deposition from solutions that were expressed upward from the batholiths as they cooled. The batholiths themselves are essentially barren except in their upward bulges, or "cupolas," on the roofs and in the regions near the warped planes of the roofs. A "dead line" may be drawn on a section of a batholith below which valuable deposits are rarely found. This dead line lies nearer the roof in its lower regions than in its higher ones. It is about three miles or a little more from the roof in the summit areas, but about one mile from the roof in the trough areas (Fig. 3). The valuable gold lodes in deeply eroded areas of granitic batholiths are

[13] For analyses see Larcombe: *op. cit.*, 195.

found therefore in and near roof pendants on the batholith and very rarely are found in the batholith more than one mile from the contact with invaded rocks.

The cupolas, or small upward bulges, are the favored positions at all places on the roofs. These cupolas may be found at any place on the roof but for the purposes of study and comparison they are grouped in three classes; namely, the summit cupolas, the intermediate cupolas, and the trough cupolas. The gold deposits associated with cupolas that are treated in this paper are those associated with trough cupolas and intermediate cupolas. The summit cupolas do not enter into the discussion.

The cupolas are generally elongated. Some of them contain valuable lodes but in many the chief lodes are in the invaded rocks near them. In a large number of gold-bearing districts a "finger" of the cupola, or of the invading rock that surrounds the roof pendant, points toward the gold-bearing area. That seems to be true in many of the areas treated herein, as may be seen by inspection of the maps presented. These include Porcupine, Kirkland Lake, Shamva, Kolar, Kalgoorlie, etc. To these the Mother Lode region of California may be added, although it was deposited in a relatively high region compared with the deposits of the shields. When the production of these districts is considered, it is an impressive list, since it includes many of the most productive districts of the ancient shields.

In certain great gold-bearing districts there is no conclusive evidence that the gold lodes lie above cupolas. Erosion is not deep enough to expose the cupolas if they exist. In other districts erosion has gone so far that the cupolas are joined to the main invading masses. In still other areas erosion has reached the stage where a study may be made of the relations of the deposits to the cupolas and in most of these it is found that the major deposits have formed in and around the cupolas. It is a rational inference, therefore, that most of the larger deposits of gold of the deep-seated type have formed in and around cupolas, and that these relations would generally be shown if erosion generally had reached the stage suitable for the study of their relations to the mineralizing masses.

Why gold lodes are concentrated in and around and above granitic cupolas is a matter of speculation. At such places fractures are likely to be closely spaced and gold veins are deposited in them. But fractures are commonly found over wide areas and relatively few are metallized with workable deposits. The fracturing is a necessary condition for the formation of gold veins but probably it is not the principal cause of their deposition. The concentration of the gold-bearing solutions in the cupola areas of the batholith as illustrated by Fig. 3 is believed to be the controlling factor of their localization in and about the cupolas or upward swells of the batholiths.

ACKNOWLEDGMENTS

The writer wishes to thank Mr. W. I. Gardner for valuable assistance in connection with preparation of certain maps that are presented with this paper, and Mr. William Applebaum for translations of Russian literature and for transliterations of certain maps.

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The Ore Deposits of the Tri-State District (Missouri-Kansas-Oklahoma) *

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Contents

Scope of This Report

The Tri-State district, as outlined in this paper, refers to the entire mineralized area in southwestern Missouri, southeastern Kansas and

* A more comprehensive paper is contemplated later.

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northeastern Oklahoma (Fig. 1). The part of the district in Missouri and near Galena, Kans., is still sometimes referred to as the Joplin district (Fig. 4). The Picher-Miami area is that part of the Tri-State district located contiguously in Kansas and Oklahoma in the vicinity of Baxter Springs, Picher and Miami.

This paper is confined almost wholly to the ore deposits of the Tri-State district and particularly to the Picher-Miami area.

Our study of the ore deposits of this district dates from late in 1925 (by Fowler) and early in 1927 (by Lyden) to the present. Our work has been confined almost entirely to the search for ore for numerous companies and individuals. It was not possible for us to undertake numerous highly interesting investigations which presented themselves but which would not immediately help in finding ore. We particularly refer to the general structural geology of the Ozark uplift, the genesis of the ore, and other problems of a similar nature. The local and general structural problems of the district are being solved gradually as the work progresses. The factors that control the loci of the several orebodies are evident.

This paper is based almost exclusively upon our own observations. Most of the important mines in the Picher-Miami area were studied in detail. A map (Fig. 5) is included which shows this area and the properties we have examined to date. When it was convenient we visited, studied and compared structural relations in numerous mines in states outside the district. Some of these observations are noted in this paper.

Production of Tri-State District

The aggregate production of the Tri-State district, as a whole, from the time of its discovery in 1848 to Jan. 1, 1931, has been compiled by Otto Ruhl, mining engineer of this city, as follows: zinc concentrates, 15,253,206 tons; lead concentrates, 2,586, 511 tons; gross value of zinc and lead concentrates combined, $799,140,477.

Figures have been compiled by J. P. Dunlop, of the U. S. Bureau of Mines, showing the production of that part of the Tri-State district known as the Picher-Miami area, from 1909 to Jan. 1, 1931, as follows:[1] zinc concentrates, 7,630,326 tons, lead concentrates 1,365,387 tons.

Historical Sketch

The first mining in what is now the Tri-State district was probably done near the present site of Joplin, in 1848. Lead was the mineral

[1] These figures probably include numerous small mines in northeastern Oklahoma outside the Picher-Miami area, the production of which was small.

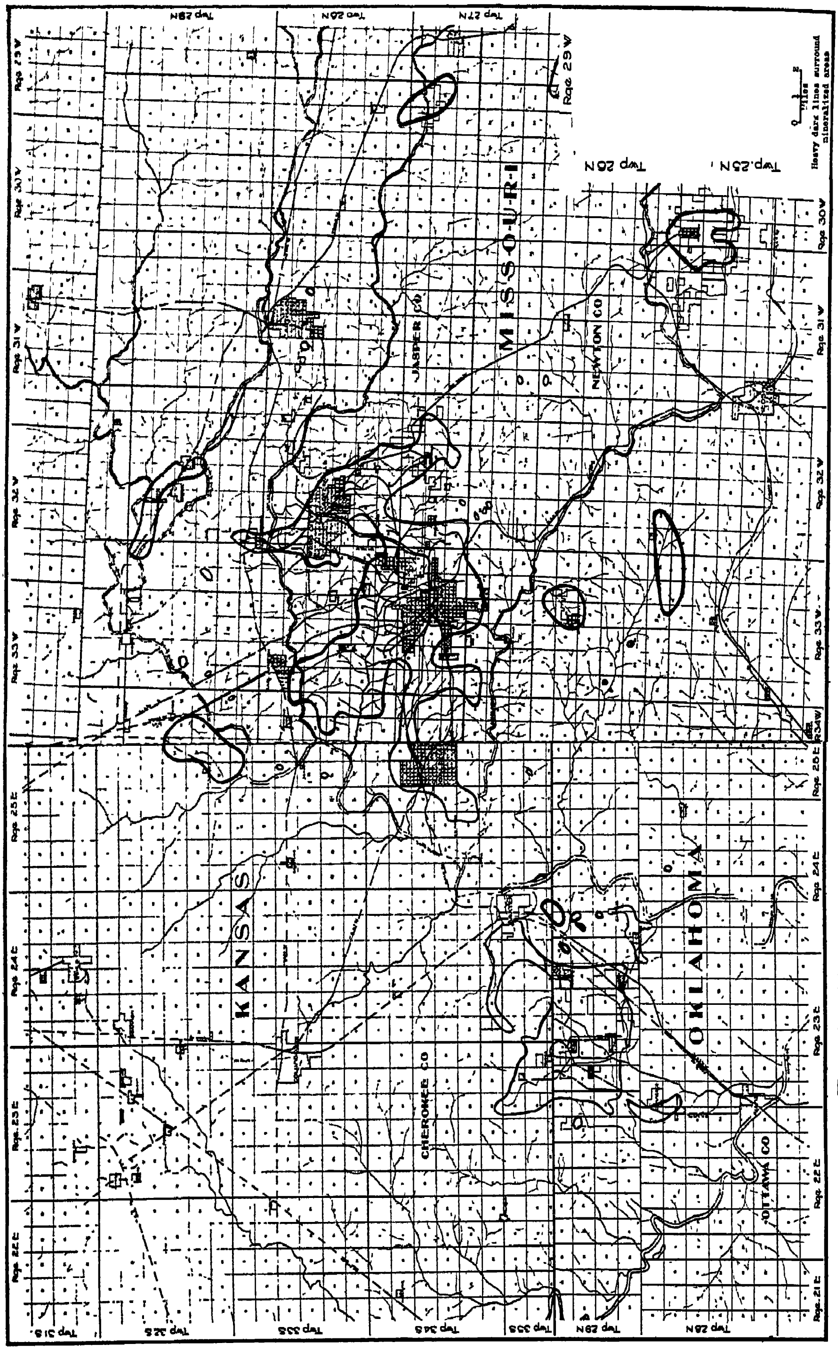


FIG. 1.—THE TRI-STATE DISTRICT (EXCEPT EXTREME EASTERN PORTION).

sought at that time. The mining of zinc followed and resulted from the mining of lead. These operations started on ore at grass roots. Since 1848, mining operations have been carried on almost continuously.

The early operations were confined to districts in Missouri. The first mining in Kansas, near Galena, was done in 1876. Important discoveries were made the following year and the Galena district soon became an important producer of lead. The first mining in northeastern Oklahoma was done in 1891, near Peoria. Discoveries of ore near Lincolnville,

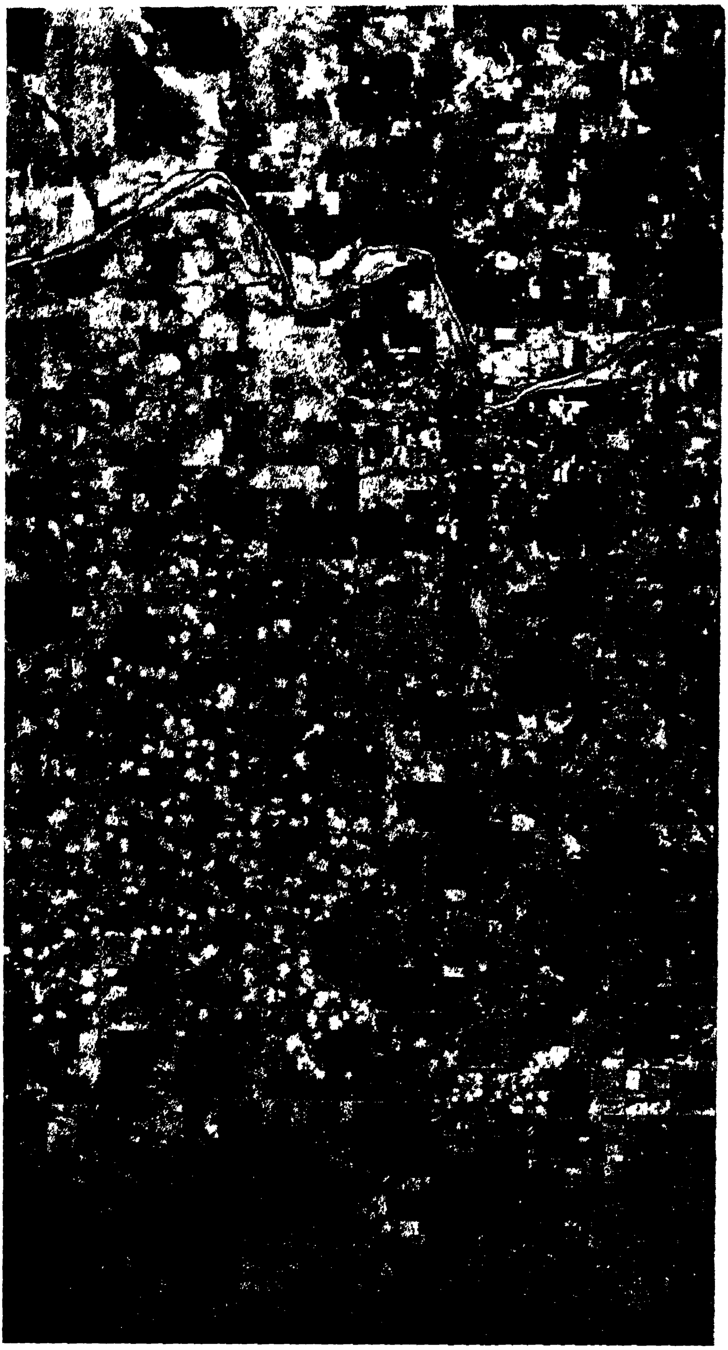
Courtesy Wingo Studio, Joplin, Mo.

FIG. 2.—AERIAL PHOTOGRAPH OF PICHER, OKLAHOMA.

Miami and Picher followed in the order named. Large-scale mining operations, in what is now the Picher-Miami area, date from about 1916.

The ore deposits in the Picher-Miami area were richer and more concentrated than those in the old Missouri and Kansas areas, consequently the mining activities after 1918 were confined very largely to the new district.

In the Waco district, on the Kansas-Missouri boundary, 18 miles northwest of Joplin, mining operations were carried on from the time of its discovery 14 years ago until June, 1931.



Courtesy U. S. Army Air Service and U. S. Geological Survey.

FIG. 3.—AERIAL MOSAIC OF OKLAHOMA-KANSAS ZINC AND LEAD MINING FIELD, SHOWING PICHER, CARDIN, COMMERCE AND OTHER TOWNS. Small rectangles are city blocks; large ones represent cultivated areas. Light colored round and irregular areas represent mine and mill dumps. Irregular line is Spring River. Scale: 1 inch = 2.14 miles.

Other areas in the Tri-State district around which mining has centered intermittently for several years are Aurora, Stotts City, Wentworth and less important points.

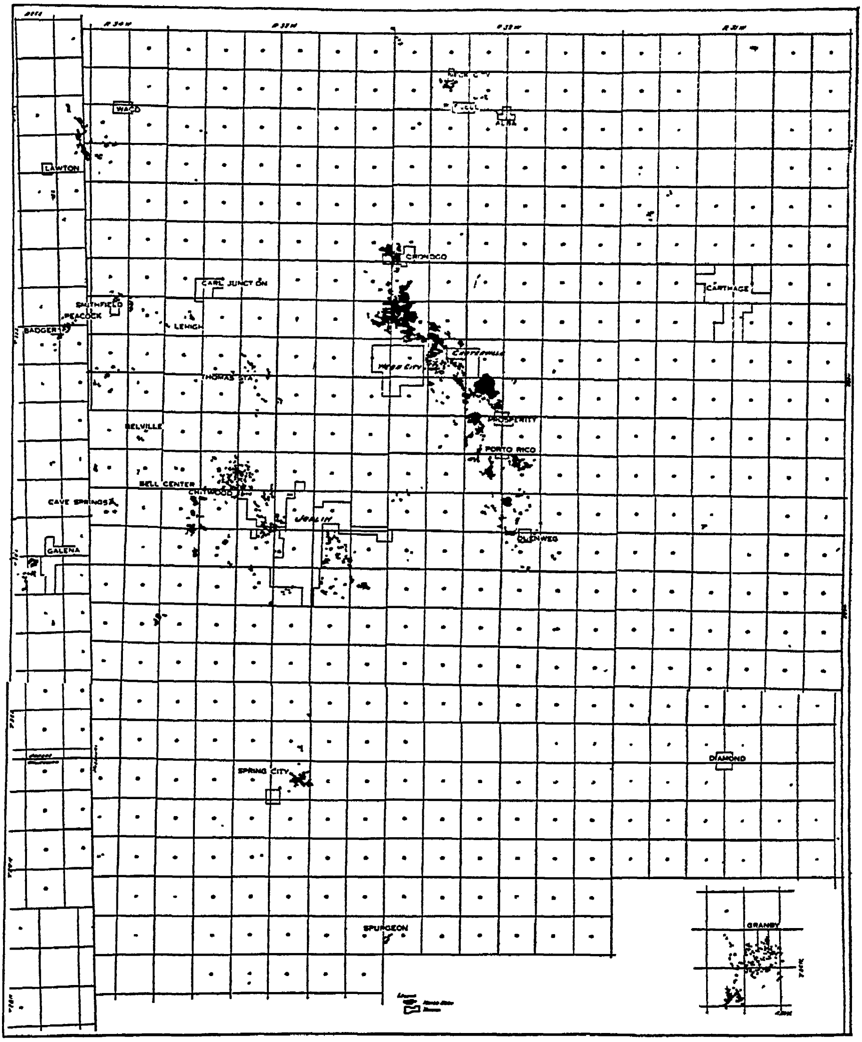


FIG. 4.—THE OLD JOPLIN FIELD.
Scale: 1 inch = 5 miles

LOCATION AND TOPOGRAPHY

The Tri-State district is a nearly continuous mineralized region on the northwestern flank of the Ozark Mountains in southwestern Missouri, southeastern Kansas and northeastern Oklahoma. Joplin and Webb City, Mo., Baxter Springs and Galena, Kans., and Picher and Miami, Okla., are the best known cities and towns in the mining district.

As outlined to date, the Tri-State district trends roughly southwesterly from the vicinity of Springfield, Mo., to near Miami, Okla., a distance of nearly 100 miles. Its width varies from a few miles to more than 30 miles.

The elevation of the district varies from 700 to 1350 ft. above sea level. The topography bordering the streams is often abrupt.

In the Oklahoma-Kansas part of the district the surface formations are largely Cherokee shale. The region is of the rolling, prairie type. In Missouri, the Boone predominates as the surface formation and the topography is more rugged. Nearly vertical cliffs from 100 to 200 ft. high are common, bordering the streams. They often follow trends of shearing and shattering.

Geology

It is not our purpose to discuss the general geology of the Tri-State region. This subject has already been ably covered by numerous geologists. At this time it is unnecessary to discuss in detail formations older than the Boone of lower Mississippian Age.

The Boone formation comprises limestone, dolomite and chert, above which are occasional patches of upper Mississippian (Chester) limestones, sandstones and shales. The whole is overlain in the western part of the district—the Kansas-Oklahoma field—with the Cherokee shale (containing lenses of sandstone) of the Pennsylvanian. In the Joplin, Mo., and Galena, Kans., fields, the Cherokee formation occurs only as scattered outliers, mainly in depressions in the old surface of the Mississippian.

The Boone formation is the only one of importance to lead and zinc mining because it contains practically all of the known ore deposits of the district. This formation, which we believe to have been originally limestone, now comprises beds of limestone, dolomite and massive chert, one or more oolite beds and numerous chert nodule beds. A detailed description of the mineralized horizons is given later (p. 218).

Conditions underground in the Picher-Miami area are similar to those observed in like strata on the surface in the vicinity of Joplin and elsewhere throughout the district. The siliceous surfaces common in the Tri-State district are brought into prominence because of their resistant character. In unaltered areas the surface of the Boone formation is limestone. This was determined by numerous drill holes and shafts in such areas.

In the old mining fields, including the Galena-Joplin-Webb City area and the fields south and east, erosion has removed not only most of the Pennsylvanian (Cherokee) formation but also 100 ft., more or less, of the Boone formation. In the Picher area this unmineralized part of the Boone, together with a varying thickness of Cherokee shale, remains in place over the ore deposits.

No pronounced unconformities were noted within the Boone formation in the Tri-State district. The numerous strata comprising the formation are recognizable in all parts of the district. Single beds or strata may be traced for several miles. In some instances, solutions have thinned beds locally.

The part of the Boone formation below M bed (p. 218) is generally more massive than the part above this bed. Granite and other igneous rocks underlie the entire sedimentary series of the Tri-State district. The known depth of these rocks below the surface varies from approximately 1200 ft. at the Bird Dog mill, west of Picher, to approximately 2000 ft., as shown by various drill holes and deep wells. A number of deep wells in various parts of the district encountered igneous rock at depths of from 1600 to 1800 feet.

Small areas of igneous rock outcrop about 100 miles easterly, southerly and westerly from Joplin. These exposures are near Decaturville, Camden County, Mo., at Spavinaw, Okla., and on Rose Dome, near Yates Center, Kans., respectively. Opinions differ regarding the age of the granite at Spavinaw. The Decaturville and Rose Dome outcrops have been definitely designated as post-Cambrian; the latter being post-Pennsylvanian. All of them have been described by other writers.

At Spavinaw we noted very small, silicified zones of shearing extending from the granite into overlying sedimentary rocks. Pyrite was in evidence in the shear zones in a few instances.

Rose Dome has been described by Twenhofel and others.[2] We visited the area. Granite outcrops at the surface and intrudes Pennsylvanian shale, which shows metamorphism in the zones of induration. Some of the granite contains shale inclusions.

The surface at Rose Dome is 1300 ft. above the top of the Mississippi limestone. The area is a typical Kansas plains country. Metamorphism has been intense in the vicinity of the intrusion. A well approximately 1600 ft. deep, in the sedimentary formations about 300 yd. northwest from the large granite intrusion, showed abundant chert in the limestone horizons.

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[2] W. H. Twenhofel: The Silver City Quartzites, a Kansas Metamorphic Area, *Bull.* Geol. Soc. Amer. (1917) **28**, 419–30.

Granite Boulders in (?) the Pennsylvanian of Kansas. *Amer. Jnl. Sci.* (1917) **43**, 363–80.

Additional Data Relating to the Granite Boulders of Southern Kansas. *Amer. Jnl. Sci.* (1919) **48**, 132.

Intrusive Granite of the Rose Dome, Woodson County, Kansas. *Bull.* Geol. Soc. Amer. (1926) **27**, 403–12.

W. H. Twenhofel and E. C. Edwards: The Metamorphic Rocks of Woodson County, Kansas. *Bull.* Amer. Assn. Petr. Geol. (1921) **5**, 64–74.

An Extension of the Rose Dome Intrusives, Kansas. *Bull.* Amer. Assn. Petr. Geol. (1928) **12**, 757–762.

Geology of the Tri-State District

A major unconformity exists between the Cherokee shale and the underlying Boone formation in the Picher-Miami district. In some places the Cherokee and Boone are roughly conformable; in others the base of the shale is nearly horizontal and the underlying Boone is flexed to

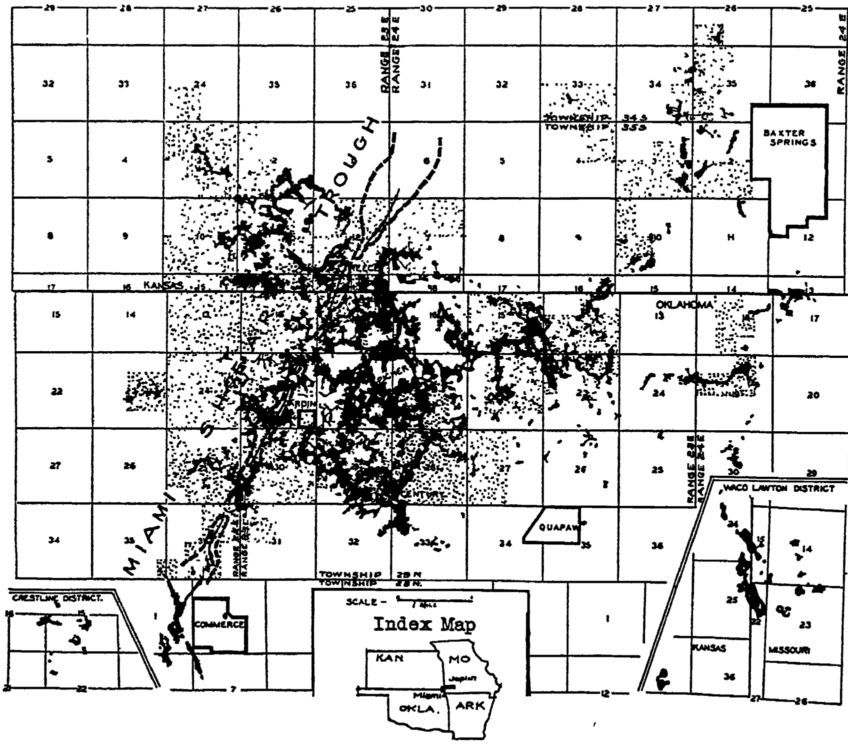

FIG. 5.—THE PICHER-MIAMI AREA.

Total area mined was 2320 acres.

Dotted area indicates portion of district studied in detail upon which this paper is based.

Heavy broken lines indicate 700-ft elevation contour on surface of Boone limestone in Miami shear trough.

a marked degree. Figs. 15 and 16 show this condition. It is certain that there was a long time interval between the deposition of the Boone and Pennsylvanian formation.

Small outliers of the Carterville remain in a few places throughout the district. In the extreme southern part of the district, near Miami, this formation (designated as the Mayes in Oklahoma) is fairly widespread.

The surface topography of the Boone formation was developed by two processes: (1) by erosion and solution of the surface before the deposition of the shale, and (2) by slumping due to metamorphism and solution of underlying limestone beds. Most of the slumping followed the deposition

of the shale and the resulting depressions are the pronounced irregularities in the surface of the Boone formation. They occur over areas of shearing and shattering where solutions metamorphosed and dissolved large quantities of limestone. In the excavation for the crusher at the Bird Dog mill the bedding of the Cherokee shale conformed with the underlying surface of the Boone formation. Both formations dip westerly into a

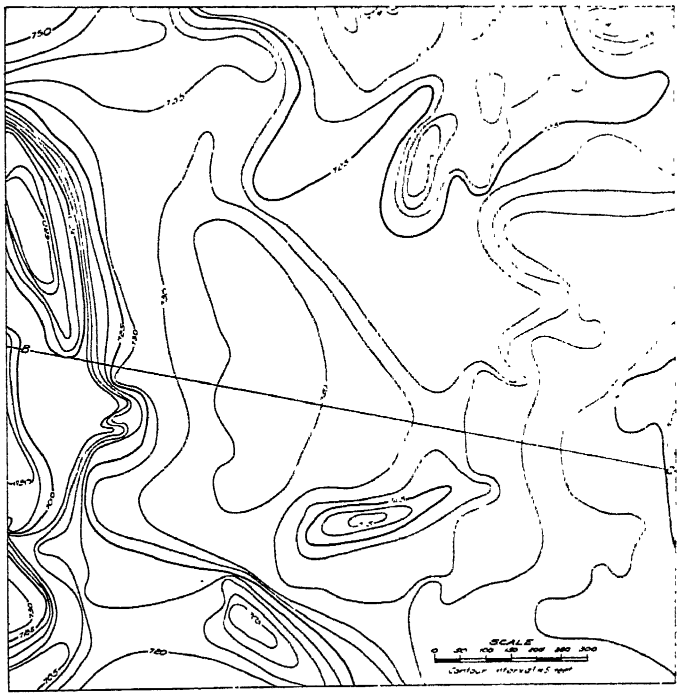

FIG. 6.—WOODCHUCK MINE. SE.¼ OF NE.¼, SEC. 30, T. 29 N., R. 23 E., NEAR CARDIN. OKLAHOMA. CONTOURS ON PENNSYLVANIAN SHALE-BOONE FORMATION CONTACT.

structural depression. Drilling across the Miami shear trough shows that the sandstone strata in the Cherokee shale conforms roughly with the depression. This proves that the depression formed after the deposition of the sandstone. In mine workings under these depressions similar effects of solution and slumpage may be observed. The so-called "karst" topography belongs largely to the first process described above.

Dolomite in three forms is widespread in the Tri-State district. The first form replaced the Boone limestone, in areas of disturbance, and preceded all sulfide mineralization. This dolomite is fine grained and gray to brownish in color. The second form is a gray dolomite of coarse texture, usually called "gray spar." It is found in association with the sulfide mineralization in many mines. The third form, pink in color, is found near the edges of the orebodies in veinlets and in crystals coating

vugs and openings in the orebodies. The first dolomite applies to any magnesium-bearing limestone.

The chert, so common in all parts of the Tri-State district, resulted from silicification of the Boone limestone in disturbed areas. This silicification was introduced into the Boone formation through zones of adjustment. Chertification instead of silicification is used to describe this phenomenon, in order to reserve the latter term for the siliceous emanations so common in connection with the sulfide ore deposits of this and other districts. Broadly, the term is meant to include all silicification prior to the introduction of the jasperoid. The chertification completed its cycle before the jasperoid and ore-bearing solutions were introduced.

Chert nodules are common in all parts of the Tri-State district. They are of secondary origin and are peculiar to certain strata. The formation of nodules is described under Metamorphism. They were formed contemporaneously with the great mass of chert found throughout the district. Limestone or dolomite nodules are rare. The several nodule beds which are almost identical over large areas are described in detail under Stratigraphy and Metamorphism.

The mineralization that deposited the jasperoid was contemporaneous with the sulfide mineralization. Jasperoid was the first material to commence deposition from these solutions. It varies in color from light gray to black. The coloring is probably due to foreign carbonaceous material from the shattered host rocks. The jasperoid cemented brecciated rock and replaced limestone and dolomite, often without destroying the original bedding or fossils. Jasperoid is distributed irregularly through the shattered or open parts of the Boone formation, from top to bottom, and is found in nearly all parts of the Tri-State district. It generally accompanied sulfide mineralization, but barren jasperoid may be found in small quantities in almost any part of the district.

Shale, which was deposited during Cherokee times, is now often found in the disturbed areas of the Boone formation in openings of all sizes and shapes and in places sometimes 300 ft. or more below the top of the Boone. In some areas it traveled laterally for long distances from the contiguous shear zones in the Boone formation, through which it was transported downward into the ore-bearing horizons. Shale precluded the deposition of ore, particularly zinc, in some zones which otherwise were favorable.

Dark, heavy, tarlike oil is found in a number of mines about Picher. Its source is thought to be the contiguous Cherokee and Boone formations. Generally it is most abundant in small shattered domes in the Boone formation, directly under the Cherokee shale. Owing to the lowering of the water table, incident to mining operations, the oil has worked down through broken ground into some of the mines. In a few of these mines the oil has been allowed to accumulate in sumps, from which it is recovered and sold to roofing companies.

Stratigraphy

A columnar section of the Boone formation is submitted herewith. If the Boone formation had not been metamorphosed it would probably have been impossible to make these minute subdivisions, as the entire thickness consisted of nearly massive limestone. The section is nearly uniform as to thickness (except where partly eroded) and as to characteristics and qualities favorable or unfavorable for ore depositions, over the entire Picher-Miami area, and probably in the Tri-State district as a whole. A general geological section in the Tri-State region is also given herewith.

General Geological Section in Tri-State Region

System	Series	Character of Formations	Thickness, Ft.
Carboniferous..	Pennsylvanian	Shale and sandstone (Cherokee) This formation covers most of the surface in the vicinity of Baxter Springs, Picher and Commerce. It is largely absent except in sink holes, in the old fields of Missouri and Kansas	0 to 250
	Mississippian	Carterville formation—sandstones, limestones and shale. Occurs as outliers in sink holes in all parts of the district. In the extrème southern part of the district between Miami and Lincolnville, Okla., this formation covers areas of several square miles	0 to 50
		Boone formation—limestone, dolomite and chert—originally all limestone	200 to 400
Devonian		Shale (Hannibal or Chattanooga). Absent in most of the Tri-State district. In drill holes a few miles south of Joplin this shale is from 1 to 6 ft. thick. The formation gradually thickens to the south and east	0 to 60
Ordovician.....		Largely dolomite	700 to 1000
Cambrian......		Dolomite and sandstone	0 to 800
Pre-Cambrian and probably more recent..		Granite and other igneous rock	

The beds comprising the Boone formation are very persistent and uniform over large areas. Detailed sections in brecciated, unbroken, siliceous or limy areas in numerous mines in all parts of the field between Baxter Springs and Miami show almost uniform thickness of the several strata comprising the mineralized part of the Boone formation. Strata

STRATIGRAPHIC SECTION, BOONE FORMATION

Bed	Thickness, Ft.	Characteristics
B	0–125 (depending upon erosion)	Gray and brown limestone and dolomite
C	25–32	Gray and brown limestone and dolomite. Blue, brown and white flint (blue flint horizon near base)
D	18–22	Nearly white limestone, dolomite and flint. "Cotton rock" horizon
E	5–8	Generally coarse, "sandy" dolomite or flint. An important ore bed in some mines. This bed is an important source of galena
F	12–15	Limestone or flint. Generally barren
G	8–13	An important ore horizon in a number of mines in the vicinity of Picher. Ore occurrences in this and H bed are generally similar but this bed is richer. G and H are often mined as one bed
H	15–20	An ore horizon in some mines. Thin-bedded limestone or cherty bands 2 to 5 in. thick with the ore gradually replacing thin (¼ to 2 in.) favorable strata. This stratified mineralization almost invariably occurs with banded galena above the sphalerite. Sometimes ore found is in 8 to 10-ft. stratum directly above J. Nodules generally absent in beds F, G and H
J	4–8	Barren limestone or flint. Mineralized only in areas of intense shattering. Sometimes occurs as soft, greenish, limy stratum
K	8–12	An important ore bed. Comprises rounded nodules 5 to 9 in. dia. in upper part of bed with long, larger nodules in lower part. Ore generally confined to the upper two-thirds of bed
L	26–30	Massive limestone or flint. Contains ore in zones of intense shattering. Sometimes a 5-ft. stratum of oolite is found near the bottom of horizon
M	19–22	One of the most important ore beds of the Picher-Miami area. Where metamorphosed, definite nodules from 4 to 12 in. dia. occur throughout entire thickness of bed. Often large nodules (from 6 to 12 in. thick by 2 to 5 ft. dia.) are found at bottom of bed. This bed as mined in different parts of the Picher-Miami area varies in thickness from 5 to 22 ft. The bottom 12 ft. is the most productive horizon
N	20–25	Massive limestone, dolomite or flint with very few large (1 ft. thick by 5 to 15 ft. dia. nodules. Sporadically contains ore. In some mines a "porcelain" stratum 1 ft. thick is found 6 to 8 ft. from top of bed
O[a]	8–9	Important ore bed in a few mines. Round, flat nodules (2 to 4 in. by 3 to 6 ft.) embedded with cherty bands 1 to 4 in. thick. In some mines contains interbedded layers of nearly pure galena or sphalerite, or both, varying in thickness from a fraction of an inch to several inches. These sheets of ore are often very persistent over large areas. Such mineralization comprised the "sheet ground" mines in the old Missouri fields. The southern part of the See-Sah mine, near Cardin, Okla., is in this bed and is in "sheet ground" formation
P[a]	8–11	Large flat chert nodules interbedded in chert. Barren in most instances. Sometimes mineralized with beds O, P and Q, making an ore horizon 30 to 38 ft. thick
Q[a]	17–18	Limestone and flint, generally massive. Only a relatively few churn-drill holes have reached depths greater than 50 ft. below Q bed of the strata.
R	55±	Largely dense limestone, dolomite or gray and blue flint, with chert nodules. The Riverside mine, on Shoal Creek, near Joplin, is in this horizon

[a] The Grand Falls horizon described by Smith and Siebenthal has not been defined definitely. It falls within the range of beds O, P and Q, described above.

in the vicinity of Joplin may be correlated with similar strata in the Picher-Miami area.

Correlation by H. S. McQueen, of the Missouri Bureau of Geology and Mines, was made from drill-hole cuttings from a well at the Bird Dog mill in SE.¼, SE.¼, sec. 13, T. 29 N., R. 22 E., Ottawa County, Oklahoma, in 1930, is included to show the formation below the Boone limestone.

STRATIGRAPHIC SECTION, BIRD DOG WELL
By H. S. McQueen

System	Thickness, Ft.	Depth from Surface, Ft.	Characteristics
Pennsylvanian	100	0–100	Shale
Mississippian	315	100–415	Chert, limestone and dolomite
Ordovician	5	415–420	Brown, finely crystalline dolomite; sand grains; brown, white and sandy chert; some dark green shale and pyrite
	5	420–425	Gray, finely crystalline dolomite; 40 per cent insoluble, sand grains, chert and shale
	10	425–435	Light brownish gray, fine-grained sandy dolomite; dark brown quartzose and glassy chert, with associated fragments of sphalerite; also gray white chert
Cotter formation	25	435–460	Dense gray to brownish gray dolomite; fine-grained sand; white, gray and brown chert; some pyrite and few fragments of sphalerite
	70	460–530	Gray, dense, fine-grained dolomite. Insoluble residues comparatively small—5 to 35 per cent; considerable sand—495 to 520 feet
	320	530–850	Dolomite similar to above, with gray and blue gray chert oolitic and banded to 585 ft.; below that depth brown chert dominant material. Shines of zinc at 680 feet
Jefferson City formation	45	850–895	Dark brownish or bluish gray, finely crystalline dolomite. Insoluble residues small except 850 to 855 ft., 40 per cent brown, sandy and quartzose chert
Robidoux formation	115	895–1010	Light gray, fine-grained dolomite; sandy in basal part, with some chert
Gasconade formation	150	1010–1160	Gray and bluish gray finely crystalline dolomite; insoluble residues up to 50 per cent carry chert
Van Buren formation	20	1160–1180	Blue gray dolomite; insoluble residues contain chert; also sand and fragments of porphyry at 1180 feet
Gunter member	5	1180–1185	Dark gray finely crystalline sandy dolomite; insoluble residues 5 per cent contains: sand, fine generally rounded and frosted grains; with chert white and sandy; also granite, bluish gray and red; both types kaolinized; pyrite
Pre-Cambrian: Granite and possibly porphyry	10	1185–1195	Granite, red and bluish gray, the latter finer grained and chloritic and possibly a rhyolite. Both carry pyrite and hornblende
	11	1195–1206	Granite, red, very coarse grained, with fragments of bluish gray like above. Pyrite

At this time sufficient evidence is lacking to determine the age of the granite found in the bottom of the deep well at the Bird Dog mill. An

analysis of the cuttings from the Bird Dog well by F. E. Gregory, of the Eagle-Picher Co., shows the granite crystals to have been distorted by flowage after primary crystallization was complete. In the cuttings from the deepest 16 ft. of the granite (from 1190 to 1206 ft. depth) he found the primary minerals, orthoclase, quartz, biotite, hornblende, rutile, hematite and apatite, and the secondary minerals, chlorite, zoisite, sericite, muscovite, quartz, calcite, gypsum, epidote, augite, marcasite, pyrite, fluorite, barite, and sphalerite. The top 5 ft. of the granite (from 1185 to 1190 ft. depth) is greatly altered. The primary minerals are orthoclase, biotite and rutile; the secondary minerals, kaolin, chlorite, quartz, zoisite, calcite, gypsum, barite, quartz (pseudomorphic after calcite), pyrite and sphalerite. The cuttings from the next 5 ft. (1180 to 1185 ft. depth) are a mixture of the highly altered granite, with dark gray, sandy dolomite, chert and pyrite.

The Bird Dog well is evidently on a granite "high," as a well on the Lucky Syndicate lease 1½ miles easterly from the Bird Dog well was drilled 1525 ft. deep and did not reach the granite.

Metamorphism

The Boone formation was originally limestone, which was deposited as conformable horizontal strata. It is probable that minute quantities of silica were deposited with this limestone. Any silica deposited at that time is regarded as original; all later silica is regarded as secondary. Few limestones are entirely free from foreign material of some kind. These minute quantities are disregarded in this paper. Our interest centers in the great areas of chert that are so characteristic in the Tri-State district. We believe all of this chert to be secondary. In color, it ranges from nearly white to dark gray. In appearance the chert varies from vitreous luster to dull. Chert is found only in areas of deformation. Its quantity is proportional to the degree of disturbance.

Dolomization and chertification were the early types of metamorphism. This metamorphism is always related to zones of disturbance. The period of sulfide mineralization with its contemporaneous jasperoid type of silicification followed the dolomization and chertification. The dolomization, chertification and sulfide mineralization probably came from the same original source and were introduced into the Boone formation through the same general zones of adjustment. Shear zones and shattered areas were particularly favorable channels.

In the Boone formation the chert beds exhibit nearly identical characteristics in all parts of the district, depending on the degree of metamorphism. (See Figs. 11 and 14.) The 100 ft. of limestone directly above the ore horizon chertified only in zones of very intense disturbance. Complete chertification embodying vertical horizons of 100 ft. or more is common in areas where strong shear zones developed. In some instances

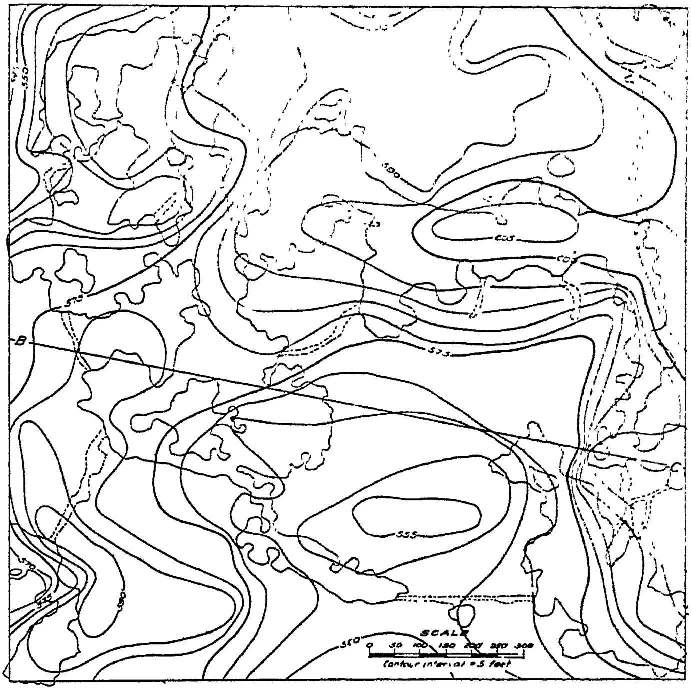

FIG. 7.—WOODCHUCK MINE. RELATION OF CONTOURS ON M BED TO MINED ORE-BODIES.

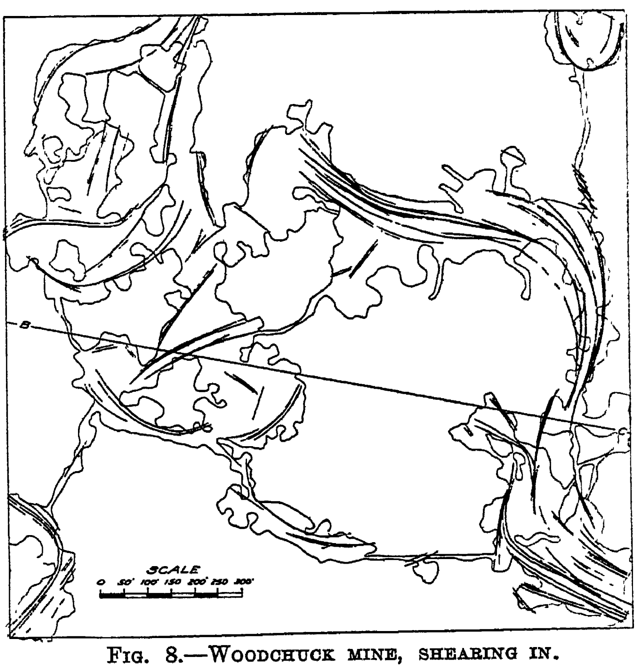

FIG. 8.—WOODCHUCK MINE, SHEARING IN.

one or more beds of the Boone formation, described under Stratigraphy, are predominantly chert over areas aggregating a number of acres. Where the disturbance was less intense less chert developed, until finally in the areas of no disturbance little or no chert is found.

Chert replaced the Boone limestone in three distinct ways, forming (1) nodules; (2) beds with cherty layers 2 to 5 in. thick; (3) massive beds up to 30 ft. thick.

FIG. 9.—M BED NEARLY COMPLETELY METAMORPHOSED.
Light colored rock is largely nodular chert; dark is largely zinc and lead mineralization.

The dolomizing solutions apparently were accompanied by sufficient quantities of silicic acid to develop chert nodules at several different horizons, the abundance of the nodules depending upon the amount of silicic acid available. Where the alteration was intense, nodule beds up to 22 ft. thick resulted. As the intensity of the alteration decreased fewer nodules developed, until finally at points distant from the zones of metamorphism no nodules are discernible. Nodules of dolomite are rare. They are sometimes found in the transition zone between the metamorphosed and unaltered parts of the formation.

The chertification naturally is more pronounced in the intensely shattered areas and in the more porous horizons, and decreases gradually with distance from these areas. In all instances any silicic acid that may have been present had a particular affinity for that part of the limestone which developed into chert nodules. These nodules, completely chertified, are often enclosed in a matrix of apparently pure dolomite.

Each limestone bed has its characteristic manner of dolomizing and chertifying. Some beds apparently dolomized or chertified readily, others only under very favorable conditions. The chert nodules have the same characteristics in each bed over many square miles. These charac-

M Bed partly metamorphosed.

FIG. 10.—M BED PARTLY METAMORPHOSED.
Note chert nodules (light) in limestone (dark).

teristics should be the same regardless of the origin of the dolomizing and chertifying solutions, because the limestone as originally deposited was responsible for the dolomite and chert nodule characteristics, whereas the dolomizing and chertifying solutions merely furnished the material.

This might be a convenient time to ask "What causes chert nodules to form in limestone and what determines their size and shape?" The limestone as deposited originally must have had characteristics, chemical and physical, that accounted for the formation, size and shape of the nodules. Certain strata in the limestone were suitable mediums for the infiltration of the silicic acid solutions. These strata vary in thickness from a few inches to 2 ft. The silicic acid solutions reached these strata through cross fractures in the host rock and often followed the strata for

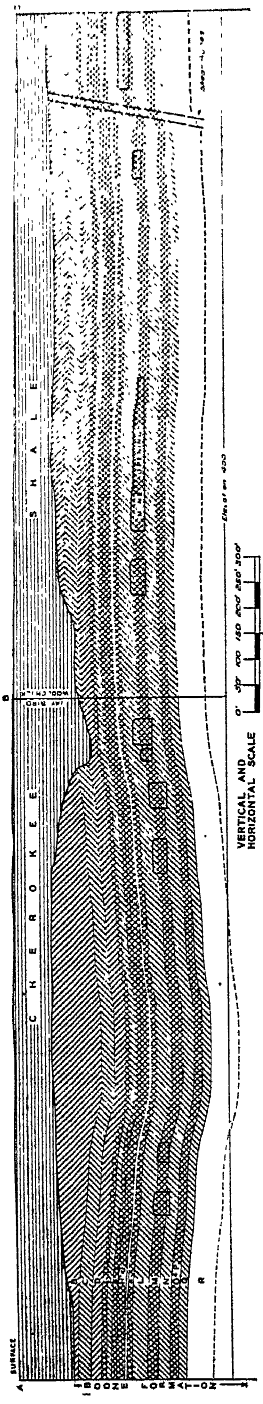

FIG. 11.—JAYBIRD-WOODCHUCK MINES, VERTICAL SECTION.
Letters refer to differentiated beds in mineralized part of Boone formation.

long distances. Concentration and precipitation of these solutions at the center of the several favorable strata gradually replaced the limestone with concentric layers of chert. The concentric bands are thought to be due to the manner of infiltration of the silicic acid solutions responsible for the chertification.

Our observations are that nodules are peculiar to certain strata; that their characteristics in the several strata are nearly identical over large areas; that they develop in an individual limestone stratum; that the thickness of the stratum influenced the shape and size of the nodule; that many of the nodules apparently developed by first forming a nucleus of dull, chalky, porous flint, known locally as "cotton rock"; that with the addition of the precipitate the cotton rock changed to dense, brittle, nearly white or gray glassy flint. Nodules in the same stratum increased in size as the concentric layers of chert precipitated, until they touched. Additional precipitation generally chertified the contiguous country rock. The phenomenon regarding the formation of nodules has been studied by the German chemist, Liesegang. Our observations, as stated above, apparently agree with some of Liesegang's findings.

Nodule beds have an important bearing on the ore deposits of the Tri-State district. The favorable association of the nodules with the orebodies was observed by the early miners, who designated them "mineral eggs." Chertification, other than the nodule type, developed by replacing completely a stratum or bed in the area of disturbance. The process often proceeded from limestone to "cotton rock" to dense flint much the same as in the nodule type of replacement.

In thin-bedded limestone horizons alternate layers of limestone were replaced by chert. In places of intense chertification all layers were replaced. In areas of less chertification the process did not go beyond the "cotton rock" stage. The tripoli deposits near Joplin are a form of cotton rock.

In parts of this district the vertical range of silicification (chertification) extends from the surface into the granite. This has been proved by the deep drill holes and by examining the granite outcrops at Spavinaw, Okla. (70 miles south of Picher). Sandy horizons in the Cherokee formation and horizons in the Mississippian and Ordovician series show this condition where shattering or porosity of the host rock made favorable reservoirs for solutions which carried the silicic acid.

To date the writers have found no original flint in the formations directly below the Mississippi formation in the Ozark uplift. It may be present in some localities. Our observations are confined to southwestern Missouri and contiguous parts of Oklahoma, Arkansas and Kansas, and to areas near Yellville, Harrison and Fort Smith, Arkansas.

In the Capps mine, at Rushville, Ark., we noted that the orebodies are in zones of disturbance in Ordovician limestone; that chertification and sulfide mineralization were most intense at the places where the limestone was sheared; and that laterally away from these places all mineralization rapidly diminished in intensity and finally disappeared where the surrounding limestone was undisturbed. This is similar to the process of mineralization in the Boone limestone in the Tri-State district.

Structure

Regional Structure

Structure has a vital bearing upon metallic ore deposits in all igneous or sedimentary formations. It is the forerunner which made the channels, through which the mineral-bearing solutions traveled, and made many of the reservoirs in the host rock in which the ore minerals were deposited. These reservoir types are described under Ore Deposits.

Structurally, the Picher-Miami area may be likened to an oil field comprising many domes, basins, anticlines, synclines and allied flexures, all in the miniature. These flexures vary greatly in size, both vertically and laterally. Deformation is more intense here than in any other like area in the Tri-State district.

In studying the structure of the Picher-Miami field we mapped orebodies, structural features and all ore-bearing horizons, both horizontally and vertically, as found in the mine workings and interpreted from drill-hole data. This involved the correlation of the ore-bearing horizons over a number of square miles and under all types and degrees of alteration and deformation.

Limited observations by us in many parts of the Ozark uplift indicate that a complete study of the area will reveal major structural features around which less important ones may be centered.

The Boone limestone (columnar section, p. 218) was classified into beds, with characteristics that made them either favorable or unfavorable horizons for ore deposition. (Description of these beds is under Stratigraphy.) The deformation of these beds, separately and collectively, was studied with the object of finding the relation between structure on the one hand and dolomization, chertification and sulfide mineralization on the other.

There are two distinct types of regional deformation in the Boone formation, characterized by (1) flexing of the beds (Figs. 7, 11 and 16); (2) a series of strong shear zones (Figs. 13 and 14). Deformation by folding caused shearing and shattering of the beds in the areas of sharp flexing. These areas were easily attacked by the circulating solutions and as a result the beds were chertified and brecciated and later mineralized in the favorable zones by sulfide-bearing solutions.

Where relief took place by shearing, and not by folding, strong shear zones resulted. Along and within the shear zones the beds were easily attacked by the circulating solutions and were chertified, brecciated and mineralized. The shear zones vary in width from a few feet to several hundred feet. Both types of deformation commenced in late Mississippian time. This deformation was followed by a period of erosion that planed off the limestone in some areas and made a fairly level surface (Fig. 15). The Cherokee shale, of Pennsylvanian age, was deposited upon this surface.

Where there is little deformation of the Boone limestone the strata are nearly horizontal. Dolomite, chert, jasperoid and ore deposits are lacking in such areas. Fig. 16, Skelton mine, shows the contour of M bed in this mine in an area where flexing is intense. Fig. 15 is a similar contour map of the same area showing the surface of the limestone. The intense flexing of the ore bed is not reflected in the contour map of the surface of the limestone. This shows, without question, that the flexing took place before the period of erosion and before the Cherokee shale was deposited.

Shearing across the ore beds produced long, narrow, high orebodies. Movement on the bedding planes characterized the sheet ground type of deposit. Often both types of deformation are found in the same area. Such zones are the loci of the major orebodies in the Picher-Miami area.

Orebodies are found in anticlines, synclines and flexures of all types. Broad and gently folded anticlines or synclines are mineralized only where small favorable reservoirs developed within them. Sharp folded anticlines or synclines are generally mineralized over the area of shattering and shearing. In asymmetrical anticlines mineralization is usually

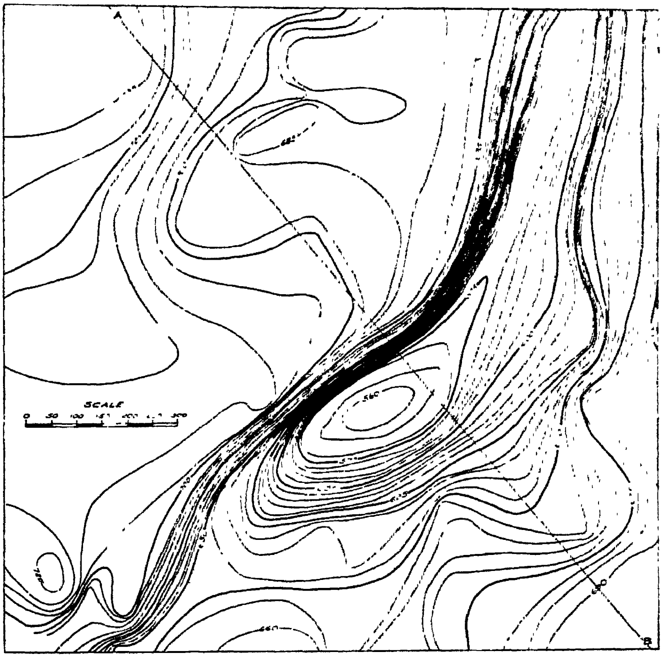

FIG. 12.—ANGORA-BLUE GOOSE MINE. CONTOURS ON PENNSYLVANIAN SHALE-BOONE FORMATION CONTACT.

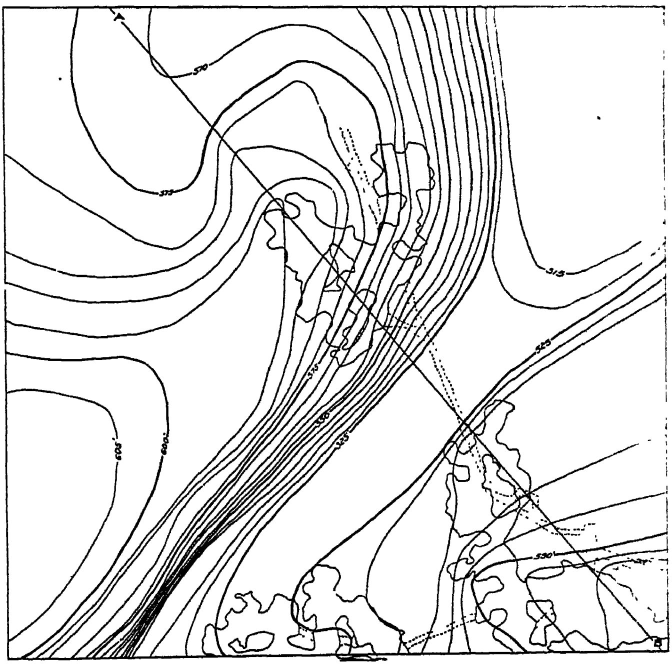

FIG. 13.—ANGORA-BLUE GOOSE MINE. RELATION OF CONTOURS ON M BED TO MINED OREBODIES.

found where shattering was intense and particularly where the steep limb begins to break over.

Deformation, by shearing, is in evidence in many parts of the Picher-Miami area. The Miami shear trough, the most pronounced structural depression in the Picher-Miami area, consists of a series of elongated basins, all within a major trough in the surface of the Boone limestone. The basins vary from 100 to 200 ft. or more in depth and from 1200 to 2500 ft. in width. They are two to three times as long as they are wide.

Another somewhat similar trough has been partly defined, by drilling, about one mile west of Melrose, Kans. Melrose is six miles west of Treece, Kans. Many drill holes in the vicinity of Melrose found lead and zinc ore of good grade at a depth of approximately 375 to 440 ft. Here the Cherokee shale covers the Boone limestone to a depth of 250 to 300 ft. Melrose is more than 12 miles from the edge of the Cherokee shale. The orebodies in this area appear to be similar to some of those in the vicinity of Picher and to be controlled by the same structural conditions. No shafts have been sunk to the ore horizon.

Following the regional shearing, solution of some of the underlying beds caused gravity tension fissures to develop at intervals across the several basins. This produced a graben-like type of structure, which became most pronounced near the middle of the basins. There was little vertical displacement at the ends of the basins.

The Miami shear trough has been traced through mine workings and by drilling, from near Miami, Okla., to near Crestline, Kans., a distance of approximately 20 miles. It passes through the mines west of Commerce, Okla., and the Scammon Hill, Roanoke, Blue Bird, Shorthorn, Angora, Ritz, Central, Beaver, Anna Beaver, Gordon, Lucky Syndicate, St. Louis No. 6, Quapaw-Chubb, Cherokee, Webber, Jarrett, Foley and Mullen mines. (See Figs. 1, 12, 13, and 14.)

Ore has been mined from strong mineralized shear zones within and along the edges of the Miami shear trough. The shear zones trend in the direction of the trough. It is our opinion that the shear zones were responsible for the formation of the trough and basins.

In the workings at the south end of the Roanoke mine and at the north end of the Scammon Hill mine there is a narrow shear zone that occupies the center of the trough that connects a basin in the Scammon Hill mine with a basin in the Roanoke mine. The trough made by this shear zone is 600 to 700 ft. wide. The shear planes are vertical, are strongest near the center of the trough, and have all the characteristics of tension shearing due to lateral stresses.

In the Scammon Hill mine the shear planes make a definite pattern with each other rather than with the basin. The northwesterly striking shear planes are arranged en echelon across the zone in a north-south direction and make connection with their complementary northeast

striking shear planes at the northern and southern boundaries of the mine. Echelon shearing is characteristic in several mines in the field.

Structure due to Metamorphism

Structure due to metamorphism is local and is of the gravity type. Dolomization and chertification were most intense in the zones of deformation of the limestone. They were accompanied by contraction in volume of the zones they metamorphosed. Adjustments took place during contraction on old shear planes around and within the zone. These adjustments in turn set up stresses at other points around the zone. These stresses were relieved by tension shearing and flexing of the beds. Intense brecciation and shattering occurred at the points of adjustment. The orebodies that conform with this type of structure follow the shearing planes and flexures.

In most instances sulfide mineralization is confined to the outer shattered edge of the zone. When it occurs more or less completely around the zone it makes the so-called "circle orebody." The Oronogo Circle mine, near Webb City, Mo., is an orebody produced by this type of deformation. Usually breccia, large and small, is found between the mineralized zone and the central core. The core may or may not be shattered.

Solution of the Boone limestone in the district has been in progress from the earliest deformation to the present. This is common with the folding type of deformation where the shattered beds allowed free circulation of solutions. Where there was solution of beds over large enough areas the weight of the overlying mass caused the block to slump. Tension shearing developed at the points of greatest flexing in and around these blocks.

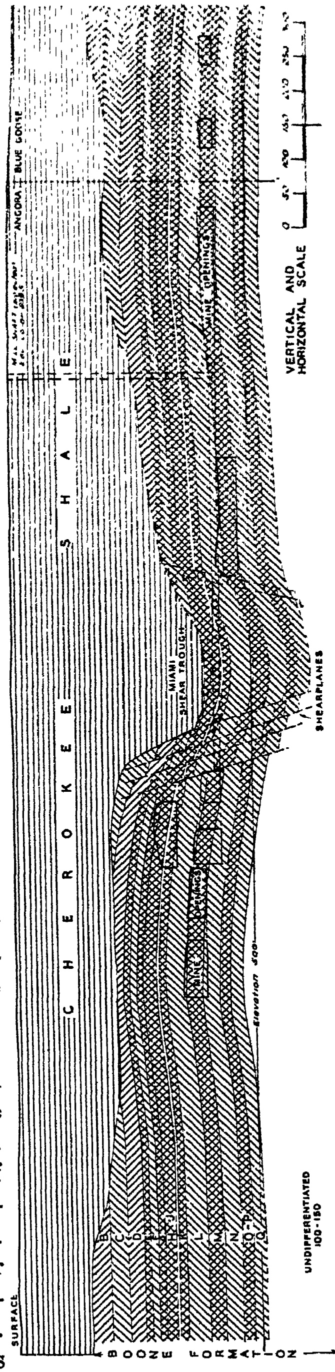

FIG. 14.—ANGORA-BLUE GOOSE MINE—VERTICAL SECTION.
Letters refer to differentiated beds in mineralized part of Boone formation.

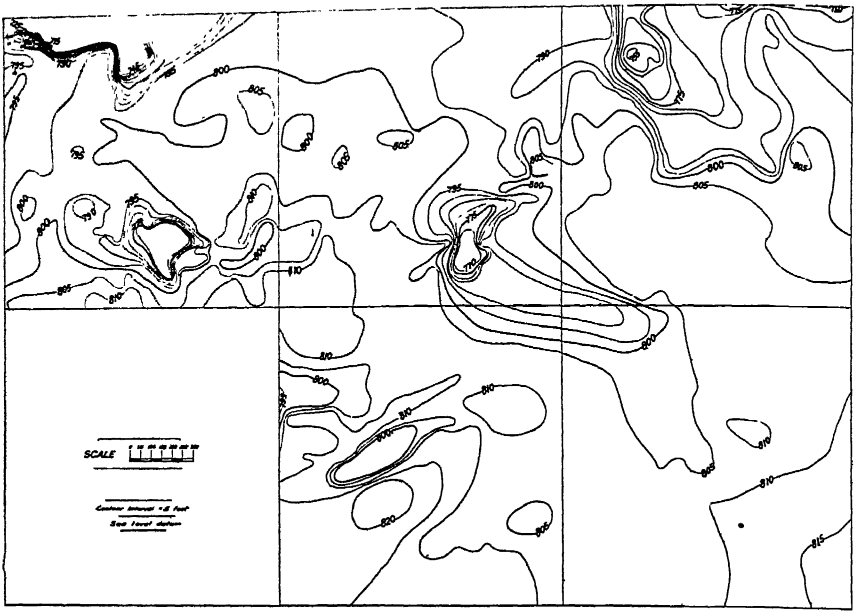

FIG. 15.—SKELTON MINE. CONTOURS ON PENNSYLVANIAN SHALE-BOONE FORMATION CONTACT.

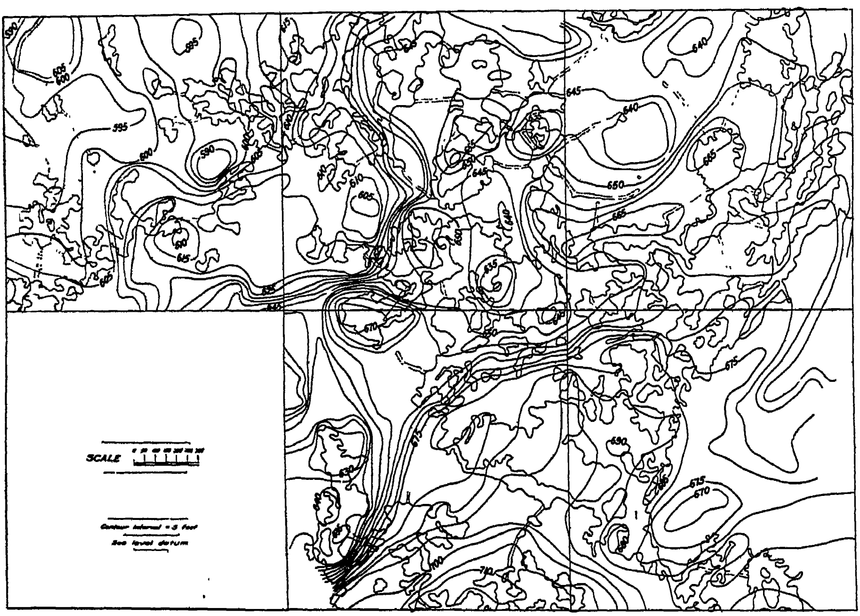

FIG. 16.—SKELTON MINE. RELATION OF CONTOURS ON M BED TO MINED ORE-BODIES.

Ore Deposits

An essential in nearly any primary, nonferrous ore deposit in igneous or sedimentary formations, anywhere, is a favorable reservoir for the deposition of the ore. This reservoir may be:

1. Porous formation that can be replaced;
2. Cavernous horizons, large or small;
3. Shattered or brecciated areas or open fissures in which the conditions for ore deposition are favorable;
4. Any combination of the above.

In the Tri-State district ore is found in these four reservoir types. Horizons of shattering account for most of the ore deposits. Zones of maximum shattering and minimum movements were particularly favorable loci for ore deposition. Often, however, the orebodies occupy only part of the shattered area, being confined to the places of most intense shearing. As is to be expected, the ore deposits are greatly influenced as to size and character by the nature of the contiguous host rock and by definite structural conditions.

The processes which controlled the formation of these ore reservoirs and made them accessible for the ore-bearing solutions are described under Structure. Stratigraphically, the ore is confined almost exclusively to six beds in the Boone formation. These beds are described in detail under Stratigraphy.

The nodules which characterize the important ore-bearing horizons vary in size from 2 to 3 in. dia. to large and nearly flat, 3 in. to 2 ft. thick and 5 to 100 ft. in circumference. The beds containing the small nodules comprise the best orebodies. One and possibly two of the ore beds seldom, if ever, contain nodules. Instead, they are made up largely of gray, coarse, sandy limestone or dolomite.

The formations between the ore beds are generally massive limestone, dolomite, or flint, depending upon their degree of metamorphism. Where shattering was intense enough these horizons are sometimes mineralized. In a few instances, as in large, long shear zones, the mineralization had a vertical range up to approximately 150 ft. There the ore is generally confined to the shear zone and did not migrate into the contiguous strata. The Pioneer mine, near Commerce, Okla., the eastern part of the Kansas Explorations Jarrett mine, near Treece, Kans., and the (East) Hartley mine, near Baxter Springs, Kans., are examples of this kind of deposit.

The ore beds vary in thickness from 5 to 30 ft. One of the most persistent beds of the district has an average thickness of approximately 20 ft. In many mines only part of the bed contains commercial ore. The mineralization in each mine is localized to the zones where structural conditions are favorable Such zones may be the top, bottom, or flanks

of domes, synclines or flexures. The beds correspond stratigraphically throughout the Picher-Miami area and probably in the Tri-State district as a whole.

Laterally, the orebodies sometimes cover many acres. The so-called "sheet ground" mines are examples of this type of deposit. There were many such deposits in the old Missouri field. The southern part of the See-Sah mine, near Cardin, Okla., and the (West) Hartley mine, and others near Baxter Springs, Kans., are "sheet ground" mines.

The outlines of the orebodies are somewhat irregular vertically, depending upon the dip of the strata and the degree and character of shattering, and are very irregular laterally. The line of demarcation between ore and waste is often sharp. In other instances it grades gradually from good ore to waste.

The loci, trend and shape of each particular orebody is determined by structure contours based upon definite markers in the ore beds. Sea-level datum is used. For areas away from underground workings the structure contour maps are made exclusively from churn-drill hole data. A knowledge of the structure can be determined within satisfactory limits from drill holes placed approximately 250 ft. apart. Each type of structure produces its peculiar shape of ore deposit.

In this district numerous attempts have been made to establish a relationship between the loci of orebodies and sink holes in the surface of the Boone formation, sometimes marked by sink holes in the shale. Several million dollars have been spent upon this hypothesis. These expenditures were made largely for drilling. Where structural conditions formed favorable reservoirs for ore deposition in the strata beneath the depressions, ore has been found, but not otherwise. It was the structure underneath, not the sink hole itself, that controlled the formation of the orebody. In the Picher-Miami area many major orebodies have been found under zones where the surface of the Boone formation was horizontal or even domed. As previously stated, structural conditions control the loci of the orebodies under *all* conditions.

Types of Ore Deposits

The important ore deposits of the Tri-State district are either hard-ground deposits or soft-ground deposits.

1. Hard-ground Deposits

1. *Brecciated.*—These are very common throughout the district. Found in or near the shattered chert, as the cementing material adhering to the chert, or as aggregates of galena or sphalerite, or both.

2. *Stratified.*—Deposits of this type may be very small or may contain many thousands of tons of ore. H bed belongs to this type.

3. *Sheet Ground.*—The ore occurs in definite, nearly horizontal interbedded sheets of (often) nearly pure galena or sphalerite, up to several inches in width. Single

sheets of ore are often persistent over areas several acres in extent. The country rock between the ore is generally barren chert, or dolomite. In the mined areas in the old Missouri field the lead to zinc ratio was higher in this type of deposit than in any other. The beds mined in these areas were O, P and Q. In the old Missouri fields sheet ground mines generally referred to mineralized strata in beds O, P or Q.

4. *In Dolomite.*—Orebodies in the dolomite (instead of chert) are rare in the Picher-Miami area. Such deposits were mined in the Douthat mine, near Cardin, Oklahoma. The orebodies at Waco, on the Missouri-Kansas line, were largely in dolomite, the chert being confined to the core of the deposits. At the Riverside mine, on Shoal Creek, four miles south of Joplin, the ore is found in a mineralized bed, 10 ft. thick, a few feet from the bottom of the Boone formation. The deposits in dolomite are similar stratigraphically and structurally to those in the chert horizons, except that the early silicification is lacking, or nearly lacking.

5. *In Shale.*—Sphalerite or galena are sometimes found in the re-deposited Cherokee shale in the deeper horizons in many mines. The aggregate ore tonnage from this type of deposit is small.

Soft-ground Deposits

Structurally the soft-ground ore deposits are identically similar to the brecciated and stratified deposits described above, except that they have been subjected to weathering. The lead and zinc ore is mixed with "mud" composed of Cherokee shale, decomposed limestone or dolomite, and "boulders" of country rock (chert, limestone, dolomite, and sandstone) of varying sizes and shapes. In the Picher-Miami field galena and sphalerite are found sparingly in transported shale. This ore was deposited in the shale during the period of sulfide mineralization. In the Joplin field this type of deposit was more widespread and was a source of ore in some mines. Such deposits are now nearly exhausted.

Soft-ground deposits often have a higher lead content than those located deeper stratigraphically, because some of the upper ore beds have higher lead to zinc ratios. In the Joplin area these upper lead beds have been eroded in part in some instances. In the Picher-Miami area (corresponding stratigraphically to those in the Joplin area) the ore beds are approximately 100 ft. from the top of the Boone formation.

In the District as a Whole

The ore deposits are very similar as to characteristics and occurrence in all of the Tri-State district that has come under our observation. All of them were controlled by the same general structural and stratigraphic conditions. Undoubtedly the general source of the ore is the same in all instances.

Ore Minerals and Analysis

The ore minerals of the district are largely sphalerite and galena. Oxidation minerals are found sparingly. The zinc and lead concentrates from the district carry only a trace of gold and from a trace to 7/10 oz. silver to the ton. The silver averages about 1/10 oz. per ton. Table 1 gives an analysis by Waring & William, Assayers and Chemists of Joplin.

The average analysis of the zinc concentrates shipments, predominantly from the Picher-Miami areas, since 1926, according to figures by Waring & Williams Laboratories, are as in Table 2. Composite lead assays by the same laboratories on 70,000 tons of lead concentrates from the Tri-State district, shipped during recent years, showed a lead content

TABLE 1.—*Analyses of Samples from Tri-State District*

Elements[c]	Sample 1[a] Per Cent	Sample 2[b] Per Cent
Zinc	58.260	59.200
Cadmium	0.304	0.058
Lead	0.700	0.293
Copper	0.049	0.054
Iron	2.230	1.900
Manganese	0.010	None
Calcium carbonate	1.880	Trace
Magnesium carbonate	0.850	None
Barium sulfate	0.820	Trace
Silica	3.950	6.971
Sulfur	30.420	31.301
	99.473	99.777

[a] No. 1 represents the composite sample of 3800 (32 tons each) car shipments of concentrates from the old Joplin district.

[b] No. 2 is a typical sample of sheet ground blende from the old Missouri fields.

[c] Nickel, cobalt, copper, gallium, thallium, germanium and indium are found in minute traces only.

TABLE 2.—*Average Analysis of Zinc Concentrates, Tri-State District*

Year	Tonnage Represented	Zinc, Per Cent	Iron, Per Cent	Lead, Per Cent	Cadmium, Per Cent
1926	440,000	58.40	1.80	0.92	0.42
1927	455,000	58.00	2.00	1.35	0.45
1928	380,000	57.60	2.50	1.75	0.56
1929	482,000	58.00	1.95	1.25	0.43
1930	308,000	58.20	1.65	0.70	0.39
1931	190,000	58.80	1.40	0.45	0.42

of 77.10 per cent. An average sample of Joplin-Miami concentrate (zinc) assays: Zn, 60.6 per cent; Pb, 0.9; Fe, 1.3; CaO, 0.8; Insol., 4.9; S, 31.1; Mn, 0.015; MgO, 0.29; As, 0.009; Cl, 0.009; total, 99.923.

Paragenesis

The paragenesis of the important minerals, as observed by the writers, is as follows:

1. *Dolomite.*—Dolomization of the original limestone. The so-called "gray" dolomite is thought to be due to a later period of deposition. A small quantity of silicic acid accompanied these solutions.
2. *Chert.*—Chertification of the limestone and dolomite.
3. *Jasperoid.*—(The first deposition from the mineralizing solutions which carried the lead and zinc sulfides.) Jasperoid is a common cementing material in the

district. The coloring comes from carbonaceous material which the solutions picked up in the host rock.

4. *Galena and Sphalerite.*—If deposited in the bedding or openings the galena is found above the sphalerite in practically every instance.

5. *Dolomite.*—(The pink spar of the miners.) Often found in connection with or close to the orebodies because deposition undoubtedly closely followed the deposition of ore minerals.

6. *Calcite.*—Filling post-mineral openings. Often in very large crystals. (The tiff of the miners.)

7. *Quartz Crystals.*—(Occur scattered or coating surfaces and small marginal deposits, or as aggregates of small clear crystals. Notably in Ballard, Hartley, Keith and other mines near Baxter Springs, Kansas.)

8. *Marcasite.*—Marcasite is found in nearly all parts of the Picher-Miami district. Sometimes it is in ore caves in large, solid sheets coating large aggregates of galena and sphalerite crystals. Here it assumes, roughly, the shape of the aggregate mass.

9. *Pyrite and Chalcopyrite.*—These are found sparingly as small crystals adhering to rocks or crystal surfaces.

10. *Sphalerite.*—Reddish crystals of "Ruby Jack" and masses rarely concentrated enough to mine. These crystals may result from the solution and re-deposition of the early sphalerite. The faces of the early sphalerite show etching in most instances.

A graphic representation of paragenesis is given herewith.

Graphic Representation of Paragenesis

Mineral	Relative Sequence of Deposition
	From first.................................to last
Limestone................	——
Dolomite.................	——
Chert....................	——
Jasperoid................	——
Galena...................	——
Sphalerite...............	——
Dolomite.................	——
Calcite..................	——
Quartz crystals..........	——?
Marcasite................	——
Pyrite and chalcopyrite....	——
Sphalerite ("Ruby Jack").	——

Conclusions

The preceding part of this paper deals largely with geological conditions as we have observed them. It is probable that other individuals having made the same observations in many cases would have reached similar conclusions. To date, the location and delineation of the ore deposits has been the problem of major importance and but little consideration has been given to the question of their source.

Structure

The importance of structure in relation to the deposits has been stressed, and the Miami shear trough has been pointed out as a major structural feature. By comparison of the Tri-State shear·zones with similar shear zones with which we are familiar, we have made certain deductions concerning them. In particular, our studies of such structures are used to interpret the *structure* of the shear zones in the Tri-State district, which exhibit many characteristics that are similar to the mineralized shear zones and fissure systems in the granite at Butte, Montana.

This analysis leads to the conclusion that, in the sedimentary formations of the Picher-Miami district, compression and shortening of the mass was lateral in a generally north-south direction, and consequently tension and elongation of the mass was approximately at right angles thereto. This type of deformation developed shear planes striking northeasterly and northwesterly and, in addition, tension fissures striking northerly and southerly. It will be noted from a map of the district that the main ore trends also lie in northeasterly and northwesterly directions.

The Miami shear trough exhibits a good example of tension type fissuring, the shear planes being arranged en echelon along the zone. The effects of torsion are evident at the points where complementary shearing relieved the stresses. If the shearing were purely by compression there would have been one long, continuous fault, with crushing along the fault plane. The strong fissures in this trough have a northeasterly trend, whereas the tension fissures across the echelon gaps between the strong northeasterly fissures have a north-south direction parallel with the direction of compression and at right angles to the direction of tension.

In areas where only the shearing type of deformation occurs, the limestone beds on either side of the shear zone show little or no vertical displacement and are but little metamorphosed. This shows that movement was confined to the shear zone and that adjustment of the blocks was by horizontal movement along the shear planes.

Genesis of the Ore

Three distinct hypotheses as to the source of the ore are stated by W. H. Emmons,[3] as follows:

1. It is held that the deposits have formed by the agency of ground water descending or moving laterally through the rocks and depositing the ores at the places they now occupy. This hypothesis, in one form or another is supported by the investigations of Whitney, Chamberlin, Keyes, Blake, Winslow, Buckley, Buehler, Grant, Cox and Miser. Bain supports this theory to account for the Wisconsin zinc deposits, but not for the Joplin, Missouri, deposits.

2. A second hypothesis states that the ores are derived from rocks below those in which they are found, that the metals were leached by ground water from the deeper seated rocks that rose as an artesian circulation to deposit the ores at the places in which they are now found. This theory is supported by Bain, Van Hise, Siebenthal and Smith and Ball. Bastin has suggested that the precipitation has taken place through the agency of anaerobic bacteria.

3. A third hypothesis, that the ores were formed by thermal waters ascending from deep sources, was advocated by Jenney, Nason, Pirsson, Tarr and Spurr.

As yet, we have insufficient data to enter into a lengthy discussion regarding the source of the ore. However, our observations in this district have impressed us with the fact that these orebodies and the structural conditions associated with them are similar in many ways to the orebodies in sedimentary rocks that are known to be in areas of igneous disturbance. Metaline Falls, Wash.; Aspen, Leadville and Redcliff, Colo.; Pioche and Good Springs, Nev., are good examples of mining districts that have some characteristics similar to the Tri-State district. We have visited all of these districts and in some of them have carried on detailed studies. In all of them the orebodies are definitely controlled by structure.

The Metaline Falls district[4] is particularly notable as an area containing orebodies somewhat similar to those of the Tri-State district, which are definitely related to igneous activity. The deposits there occur in Paleozoic limestone and dolomite in zones where the beds have been intensely shattered and silicified. The average assay of these ores from several mines shows combined lead and zinc of from 6 to 13 per cent, silver less than 0.3 oz., and gold about 20¢ per ton. Igneous rocks observed within the district are a number of small basic dikes which are later than the mineralization. A large area of intrusive granite occurs about seven miles west. Amphibole asbestos in veinlets in the silicified limestone of the district attests the relationship to igneous activity.

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[3] *Econ. Geol.* (1929) **24**, 222, 223.

[4] Chester H. Steele, Butte, Mont., gave us recent data, which are incorporated in this paragraph, regarding the Metaline Falls area.

In the Boone limestone, dolomization or silicification, or both, preceded the deposition of the ore. This is similar in many ways to the process of hydrothermal alteration that is known to have been associated with orebodies of igneous origin. This metamorphism of the Boone is similar in many respects to the metamorphism of the limestone in some of the districts named above.

In the Tri-State district the great masses of dolomite, chert and sulfide mineralization could not have been derived from the halo surrounding these masses because the rocks in these halos are but little disturbed or changed, either physically or chemically.

The following, by Siebenthal,[5] concerning a large area south of the district, was offered as evidence that the orebodies were deposited by ascending meteoric waters:

> Throughout the area underlain by the unbroken sheet of Chattanooga shale there are no deposits of ore, dolomite, or jasperoid. On the other hand, along the course of the Seneca fault, northeast and southwest of Seneca, Mo., there are deposits of ore, dolomite, and jasperoid as far as 8 or 10 miles within the area underlain by the shale, the solutions having ascended through the fault zone.

This evidence applies equally well to ascending waters from an igneous source.

The fact that the cycle of metamorphism and mineralization was an orderly one and essentially stopped after the period of mineralization shows that it was controlled by conditions that do not exist at the present time. This cycle, starting with dolomization and chertification of a relatively pure limestone, was followed by a period which included silicification (producing jasperoid) accompanied by the deposition of practically all of the galena and sphalerite of the district. Following this, the cycle was completed with the period of deposition of the pink dolomite, calcite, quartz crystals, marcasite and small quantities of pyrite and chalcopyrite. Mineralization after the completion of the cycle consists essentially of the calcite and the small sphalerite crystals known as "Ruby Jack." Occasionally small galena crystals are found in the calcite. The "Ruby Jack" is possibly a redeposition of sphalerite that was taken into solution by meteoric waters, as throughout the district the earlier sphalerite shows etching.

This succession of periods of mineralization has not been repeated. There is a repetition of the sequence of mineralization within any one period, but not of an early period on a later one. The evidence is conclusive that the cycle was controlled by conditions that were characteristic of the respective periods, and that these conditions changed at the end of each period.

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[5] C. E Siebenthal: Origin of the Zinc and Lead Deposits of the Joplin Region. U. S. Geol. Survey *Bull.* 606 (1916) p. 15.

Future of the District

Mining has been in progress from numerous widely separated fields in the Tri-State district for more than 70 years. As noted earlier in this paper, the aggregate production of lead and zinc has been very great. From the relationship of the ore deposits to the general geological features of the district, it is reasonable to suppose that other well mineralized areas may be discovered within or contiguous to the area.

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Mr. M. H. Gidel, of Butte, Mont., our coworker for years in many mining districts in the Western States, spent several months with us in this field in 1927 and aided materially in the solution of the early problems.

We wish to thank Messrs. M. D. Harbaugh of Miami, and C. A. Hansen and J. P. Dunlop of Joplin, for criticisms and suggestions regarding this paper.

DISCUSSION

(*C. K. Leith presiding*)

R. M. Atwater, Jr., New York, N. Y.—I want to go back to my research work in my student days and recall some of the results I accomplished then. We were working on magnesia and magnesium chloride. We found that the solution of magnesium chloride had some remarkable qualities. The permeability of that solution is noticeably stronger than that of any of the allied solutions. Our experiments took this form: In a box or tank, arranged for osmosis, we put a diaphragm of porcelain. On one side we put clear, distilled water and on the other side various solutions. In the end we put a mixture of solutions. For the final experiment we had chloride of magnesia, chloride of sodium and chloride of potash and sodium silicate. We found that the osmosis showed that the magnesium chloride went through the porcelain

membrane about 10 times as fast as any of the others, so much so that we got the osmosis balance of the magnesium chloride almost before any of the others started to go through. An amusing illustration of that was afterward made in the classroom. We filled a winecask with water until it was watertight, then emptied it and filled it with magnesium chloride solution, which leaked out in a short time.

What is the application of that to this situation? I remember finding in the bottom of the Jasper mine out here near Joplin a stope 20 to 30 ft. thick under Pennsylvanian sheet ground in dolomite. It contained blende, and the blende was disseminated through the dolomite in a way that looked so much like the dolomite ores of the Mascot mine in Tennessee that except for a slight difference in color one could not tell them apart. I do not remember having ever seen anywhere else in the field a stope in the dolomite below the sheet ground. I do not know whether that was the only one or not but in that one we found blendes, not high-grade blendes, probably not over 1 per cent, but I can imagine easily that solution coming from the blende-bearing dolomite underneath the sheet ground might easily have carried the blende upward into the stratified and shattered ground above it.

H. F. Bain, New York, N. Y.—Mr. Fowler has, as I see it, made two important contributions upon which we can all agree. The first is that he has worked out a detailed section, bed by bed, through this formation which has puzzled us all for so long. We have known the general characteristics for a great many years, but it has been extremely difficult to recognize any specific bed 10 ft. thick or 5 ft. thick and trace it through for any distance at all. So far as I know, Mr. Fowler and his associates are the first who have been able to do that consistently and check their results over the whole district. That, of course, is a matter not only of stratigraphic importance but of immense economic importance.

The second thing is that it is definitely tying up the actual occurrence of the ores to a series of fractures or faults, or whatever you want to call them, breaks in the rock. In the past we have perhaps missed some of that because on the one hand we always thought of faults as being straight lines and did not recognize the fact that rocks break largely along curves. Possibly, too, we overemphasized solutions and did not recognize the fact that the solution of these rocks would take place mainly along or near the fractured planes, and that it would be the location of these that would be important and not the solution of the rock, which was an after result.

Thirty years ago, spending one summer here attempting to do a great deal in a short time, I came to some fairly definite conclusions about the source of this material. I am not going to attempt today, after a lapse of 30 years and with the small opportunity I have had to review the situation, to put my opinions against those of these gentlemen who have studied it in so much greater detail. I can only say that I am very grateful to them for the wealth of material that is placed before us for study.

G. M. Fowler.—It is our opinion that every orebody in the Picher-Miami field is definitely related to structure. We have found no exception to this statement in nearly six years of extensive study with these ore deposits. Each type of ore deposit, large or small, fits into its definite structure and had its definite type of reservoir for the deposition of the ore from solutions. If the reservoir was a particularly favorable one the ore will be high grade; if less favorable, the orebody will be of lower grade. All these characteristics can be determined, very largely, from logs from churn-drill holes.

C. K. Leith, Madison, Wis.—Mr. Fowler, do I understand you to believe that the Grand Falls chert is the result of this deformation of the district?

G. M. Fowler.—Yes, sir. The Grand Falls chert is found only in areas of deformation and resulted from the chertification of the Boone limestone in that particular area.

Practically any zone that offers favorable ingress for silicic acid solutions may chertify or silicify. This zone may be lateral or vertical and may be on the surface or far below it. Old erosion surfaces, whether near the surface or many feet below, were particularly favorable horizons for chertification of large areas laterally.

C. K. Leith.—I am not quite clear yet about the Grand Falls chert. Do you think it is the result of a great horizontal movement of the earth's surface over a horizontal surface, or does it represent partly an erosion surface?

G. M. Fowler.—Instead it represents an area of disturbance at a particular horizon. The geologic column comprises the same horizons with practically the same thickness for each horizon in either chertified or unaltered areas. The Grand Falls chert is similar to other horizons in the Boone formation except that the silicic acid solutions sometimes traveled farther laterally in his horizon.

C. K. Leith.—Following it one step farther along, that would involve the movement of a great horizontal sheet. How does that fit in with your picture of shortening from a north-south direction?

G. M. Fowler.—We have not had a chance to study the entire Tri-State district in detail. The shortening in a north-south direction applied particularly to the Picher-Miami area. More work is necessary before it would be possible to determine the structural relations of the Tri-State district as a whole.

J. P. Lyden.—I should like to add a word on the Grand Falls chert. In areas of intense shearing and shattering, the Grand Falls chert horizon is completely silicified or chertified. In areas of little or no shattering this horizon is limy. The silicification or chertification in the Grand Falls chert horizon bears the same relation to the structure as chertification in the other beds bears to that structure. However, some horizons, and notably the Grand Falls chert horizon, permitted solutions to migrate greater distances laterally than did other horizons, and for this reason were chertified more extensively laterally.

C. L. Dake, Rolla, Mo.—I have had the opportunity in the last 15 years to study extensively the older formations of the Ozark region. I cannot say anything definite about the Grand Falls chert, for instance, but I have been engaged particularly with the problem of the chertification of the older formations.

I think we can say with a great deal of assurance that we know in our older formations of no single horizon that has been chertified for any considerable distance underground, except along definite unconformities. Over and over again we can follow a chertified horizon along the outcrop for a distance of several hundred feet in some cases, and for some miles. Still, when we trace these zones back through recent quarryings we find that if we go back 100 ft. into the hillside the chertification has almost completely disappeared; that there is practically none at all.

I have studied cuttings from many drill holes that have gone down through the formations within a few hundred yards of their exposure. On the hillside where they are exposed we can see that they are extensively chertified, but underground there is very little chertification. It is true that we do find cherty zones at considerable depths underground, and running for considerable distances with a fair degree of continuity, where they are along old unconformable surfaces. In other words, that

particular chert was formed in a preceding erosion period, at a time when that horizon was exposed to atmospheric agencies.

For instance, let us take the case of the Potosi formation in southeastern Missouri, which has so much of the rather remarkable druse, that is not only chert but covered with a considerable formation of thin layers of actual quartz crystals. Such material has been studied considerably and particularly Spurr has considered it definite evidence of hot water action in those formations. We have done a great deal of highway work in that part of Missouri in the last few years, and I have watched the road cuts through the Potosi formation. In many places it could be noted that a certain bed of Potosi dolomite was full of massive druse at the surface. If the road was cut back for any distance into the hillside, if the cut was of any great depth, the individual beds were revealed as practically solid dolomite, with no appreciable amount of druse in it.

In southern Missouri, in many places our formations, on erosion surfaces, contain masses of chert up to 10, 15, 20, 30 ft. in diameter, and occasionally I have found boulders of chert on erosion surfaces as much as 40 ft. in diameter. They were single bodies of solid, massive chert. Nobody ever drilled underground through such massive chert; it is utterly out of the question. We have hundreds of drill holes that have penetrated these cherty formations and none has ever drilled through a mass of chert of such character.

It seems to me that proof is absolutely assured that most of these cherts have been formed as definite weathering phenomena. The silica is present as disseminated material in the formation. Mr. McQueen, of the Survey, has done a great deal of work on well cuttings. Nowhere in that work did he penetrate any large chunks of chert. Certainly the finely disseminated chert has gone into solution during the weathering process and has been concentrated as the waters have evaporated on the surface of the rock.

There seems to be no getting around the fact that we have millions of tons of silica that has been dissolved in the Ozark region by ordinary circulating cold surface water. It has been precipitated, some of it in the form of crystalline quartz. In the Potosi formation, in the chief barites region of Washington County, many of these masses have developed a growth of barites with this druse. Nobody has ever drilled anywhere through a mass of barites like that. We get a growth of barites with these quartz crystals or with the chert masses. They have developed more or less simultaneously and there seems to be ample reason to believe that in this same aggregation of material cold circulating surface waters have also been able to dissolve such an insoluble substance as barium sulfate, and reprecipitate it at the erosion surface.

I want to emphasize that fact particularly, because sometimes we are told, concerning ore deposits, that certain of these bodies could not possibly have been made by cold surface waters because they have moved things that are so insoluble. Nevertheless, over and over again, through almost every formation from the top of the Cambrian up to the top of the Canadian; through the sequence, in other words, that you see in that cross-section up there, to the base of the Mississippian in this district; through the entire sequence we have many, many cherty formations. Those formations are not cherty in the sense that we see the chert underground, except along old erosion surfaces. It is finely disseminated.

I would also say this: Certainly the same process of chert formation went on in past geological periods when those same formations were exposed on the surface of the ground, because in places where the Gasconade dolomite overlaps the Potosi formation unconformably, we can find typical pieces of druse, water-worn and rounded, enclosed in the basal conglomerate that forms the lower member of the Gasconade. That process is still going on, as is shown in many road cuts in the Ozark region, which can be seen today.

M. M. Leighton, Urbana, Ill.—I believe that Dr. Dake is right in his statement that the matter of the solution of silica by ordinary meteoric water has been underemphasized in the past. In the study of the weathered zones of the glacial drift of Illinois we have found that that process is going on. As in the case of glacial drifts, which are very young formations, they are heterogeneous in composition. They contain limestone, shale and sandstone, and the various varieties of igneous rock and metamorphic rock. We find that one of the first steps in the weathering of these glacial deposits is that of oxidation and hydration, secondly solution of the limestone. We find, therefore, a zone that is free of limestone. Upon closer examination we find that in the upper part of that zone the drift has lost a part of its granitic pebbles, and furthermore, that the chert fragments are fewer and smaller than those in the lower and unweathered portion or in the calcareous portion below.

In other words, it appears that after the limestone has been dissolved out there is left that mantle which we have heretofore referred to as the more or less insoluble materials. But they are not insoluble. The silicates are subject to dissociation and the chert is subject to solution, and our present topographic studies of the underlying materials show that the quartz grains in the clays have been built upward by terminated crystals of quartz.

It seems to be very apparent that the solution of silica from the weathered zones of the land surface is going on all the time. When we consider the physiographic history of the Ozark region, the development of the successive peneplains across the various formations, the immense amount of time that has been involved in the development of those peneplains, and the immense amount of material that has been dissolved, it is no surprise that these limestones should contain so much chert in their weathered zones.

S. Weidman, Norman, Okla.—I have been working on a report of the Miami-Picher district over a period of several years and it is now about ready for publication. My interpretation of the structural features and the origin of the ore is essentially the same as Mr. Fowler's. I believe that the ores have their source from hot water solutions from below, and that not only the ore but also the dolomite are more reasonably explained as of hydrothermal origin than as cold-water minerals.

The prominent structural feature referred to by Mr. Fowler as the Miami shear zone, I describe as the Miami syncline. We are in essential agreement that it is a pronounced structural feature due to folding, fracturing and shearing rather than a solution trough due to the work of ground water.

C. K. Leith.—As a structural geologist, I should like to ask one question of either Dr. Weidman or Mr. Lyden. You talk about that shear zone and the north-south tension cracks resulting from the same movement. What are your criteria for identifying any one of those cracks as either shears or tensions in the field, aside from the general set-up and direction?

J. P. Lyden.—In the Blue Bird mine, one mile southwest of Cardin, Okla., the shear planes have a north-south strike. One of the orebodies is in a long narrow block where the formations dropped vertically about 20 ft. on north-south tension shear planes, producing a graben type of structure. This part of the structure we have interpreted as due largely to tension. Similar structures are common in many parts of the district.

In the Scammon Hill mine, one mile north of Commerce, Okla., the series of northerly striking orebodies arranged en echelon across the area between the northeasterly striking orebodies to the north and south, respectively, is also characteristic of tension. These orebodies follow definite zones of shearing. The tension shear planes are closely spaced and parallel. In places the individual shear planes in the

zone are arranged en echelon much the same as the zones are arranged in the group. We have interpreted this structure as due largely to tension caused by cross bending.

The shearing and fracturing that produced the Miami shear trough, particularly that part of it south of the Blue Bird mine, is similar in many ways to shearing and fracturing in the "horsetail" ore area which is part of the east-west age vein system at Butte, Mont. The intense fracturing in the "horsetail" zone at Butte is due to cross bending. In the Tri-State district the fracture systems and patterns have the characteristics of this type of deformation that we have interpreted as due largely to tension. In areas where flexing of the strata can be mapped in detail, it is usually easy to determine the relation between the flexures and the fractures.

Dr. Buehler mentioned* that tension fracturing was caused by normal faulting. It is not our conception that the big continuous shear zones in this district were caused by normal faulting. We believe they were caused by shortening of the area laterally in a north-south direction and elongation laterally in an east-west direction. Theoretically the direction of the force that caused this deformation is north-south and roughly bisects the acute angle made by the north-east and north-west shear zones. The movement on the north-east shear planes was laterally to the left, and on the north-west shear planes was laterally to the right.

C. K. Leith.—Can you identify a particular movement in any segment of the vein in a mine?

J. P. Lyden.—No, not lateral movements. Vertical movements are easily identified from the vertical displacement of the beds. Judging from the character of the shearing, there was very little movement on the individual shear planes but the total movement on the many shear planes in a zone of shearing may have been large.

C. L. Dake.—In any consideration of structure in the Ozark region, one must take into account more than has ever been done before the configuration of the pre-Cambrian land surface on which these sediments were laid down. Recently I have been studying southeastern Missouri and the area of pre-Cambrian rocks. We have found that they were massive, of a type that would not produce linear elements in topography. Nevertheless in places there are in them straight scarps as much as three or four miles long, which would not vary more than 200 yd. from a straight line. This indicates almost certainly that there are prominent faults in these pre-Cambrian igneous rocks. Some of the faults, the throws of which are as much as 1000 ft., are still represented in the pre-Cambrian. Cambrian rocks are deposited against the base of those old scarps, which are almost certainly pre-Cambrian in origin. The Cambrian material has been laid down absolutely unfaulted, showing that there has been no movement along those pre-Cambrian faults since the sediments were laid down against them.

We had a relief of close to 1000, or possibly as much as 2000 ft., in that pre-Cambrian land surface on which these sediments were laid down, and there would be bound to be reflected in the sediments a great deal of structure that is purely the result of deposition around those old slopes.

Anyone who is familiar with the work that has been done in recent years in the oil fields in connection with the matter of reflected structures should be thoroughly familiar with what I am talking about. A great many men have studied that condition carefully in the oil fields and are convinced that much of the structure in Kansas and Oklahoma that has been responsible for oil accumulation is related to the topography of the older erosion surfaces.

Such a condition is particularly brought out in southeastern Missouri where structure after structure can be proved to be related to those old surfaces. Presumably they occur over a large area. I see no reason why most of the structures of

* Paper presented but not published.

this region may not be closely related to the topography of that old pre-Cambrian erosion surface.

R. M. Atwater, Jr.—I did considerable mining in what is called the Einstein silver mine. I mined a lot of tungsten there during the war. There was one very good fissure and several smaller ones. I do not know why that should be the only place in the granite where there are such fissures. I do not see why such fissures might not be found, theoretically, under the entire area covered by the Boone formation. I have always felt that we were approaching this problem from the wrong point of view. Both Dr. Buehler and Mr. Fowler have taken the structure situation as it stands and have tried to analyze it. I should like to hear what causes the fracturing. I should like to know why the Boone formation should be broken up like a pane of glass dropped on the pavement. If the cause of the fracturing can be discovered, I believe the source of the mineral will be clearly indicated.

C. K. Leith.—You are asking a very difficult question. I may say, Mr. Lyden, commenting upon your explanation, that I have given a good deal of time in various districts, including the Butte district, in trying to satisfy myself as to the difference between the tension and shearing fractures. I have tried in this district. I think the criteria are extremely difficult. I realize that you have certain plausible set-ups of directions and patterns. It is a matter of specific information as to whether there has been or has not been a differential movement and its direction, which in the nature of the case is exceptionally difficult to put together, not that I have any definite reason for doubting your presentation.

G. M. Fowler.—The vertical movement on a shear plane is sometimes very small. In the district as a whole this movement varies from a few inches to many feet. This displacement can be measured accurately in all instances. At this stage of our work it is difficult to find criteria upon which to base the amount of lateral movement. When our work is assembled, in order to deal with the field as a unit, we will probably have a better idea of the lateral movement.

The single shear plane often has a width no greater than a piece of paper. Its length may be only a few inches, or a number of feet. A shear zone is often many feet wide and is made up, both in width and length, of innumerable shear planes. The larger shear zones may be traced long distances—several miles in some instances. Shear zones vary greatly in strike. Some trend in a nearly straight line over long distances; others are curved; a few are nearly circular. They also vary greatly in dip.

C. K. Leith.—Are you not using the term "shear" as practically synonymous with fissuring?

G. M. Fowler.—We are using it to denote movement.

C. K. Leith.—Even though the movement were normal to the surface?

G. M. Fowler.—We designate as shearing any movement on a plane that makes an angle with the bedding.

C. K. Leith.—A movement parallel to the fissuring, then, is shearing?

G. M. Fowler.—Fissures, such as we find in the mining districts in the western states, are rare here. Instead, adjustment has taken place along zones of shattering, which generally comprise innumerable shear planes or shear zones. Such zones might be designated as shear zones or fissure zones, although we generally think of a fissure as a plane of adjustment with a definite strike and dip. A single shear plane

often is of small dimensions both laterally and vertically, but the shear zone of which it is a very small unit may often be traced long distances.

C. L. Dake.—In that case, you would classify as shearing movements any movements that resulted from the unquestionable settling of the rock mass?

G. M. Fowler.—Yes. The adjustments due to settling of the rock mass would be shearing due mostly to tension. Such shearing is in evidence in many places underground and on the surface throughout the Tri-State district. Structural conditions underground are similar to those on the surface in all instances.

Most streams in the district follow old shear zones because erosion was more rapid where the formations were shattered than in the unaltered limestone. Such shear zones are in evidence on both sides of Shoal Creek south of Joplin.

T. T. Read, New York, N. Y.—I am not yet clear as to your concept of shearing. In the engineering laboratory we put a thing in tension and pull it apart, but when we shear a thing we push one part past another. Your getting shearing as the result of tension has me confused.

J. P. Lyden.—That is rather a difficult thing to explain exactly. We have used the term "tension shearing" to designate shearing caused partly by tension. The forces that cause shearing in the majority of cases are not due to compression alone. The element of tension is generally present; the tensional force making an angle usually greater than 45° with the shear planes. This, we believe, is the reason why it is often so difficult to determine whether a fracture is due to shear or tension. Those shear planes which have been caused by a tension force working in combination with a smaller compression force, we call tension shear planes; those caused by compression working with a small tension force we call shear planes; and those caused by compression alone we call compression shear planes. When a rock mass is pulled apart by tension we term the fracture a tension fracture. If you pull a rock apart by tension it will break at the points of least resistance and produce a rough surface unless it is an absolutely homogeneous substance. If you break a rock by compression in such a manner that relief can take place in only one direction, it will shear on a definite plane that bears a definite relation to the directions of compression and relief. It is our belief that when a rock mass is subjected to tension and compression at the same time the tension force tends to pull the mass apart at the points of least resistance, whereas the compression force tends to shear it along a plane that is determined by the direction of compression and the direction of relief. It is the combination of these two forces that produces tension shear planes. The compression force controls the fracturing and produces a smooth shear plane in material that is not homogeneous. The same material would not produce a smooth plane if pulled apart by tension.

T. T. Read.—You observed that in the Einstein mine?

J. P. Lyden.—Yes, in many mines.

M. D. Harbaugh, Miami, Okla.—I have discussed this question of shearing with Mr. Fowler, and I knew that the question would be raised as to just what he meant by shearing. I take it that what he observes as "shearing" is a fracture system which is due to rotational stress, or, in other words, one that is produced by differential movement. Of course, such deformation entails in it both tension and compression. Breaks will occur along lines of least resistance, and naturally, once a break starts, the structural deformation that follows it is due to the resolution of the forces, depending upon the various physical conditions that exist in the rocks in the area where the break occurs.

I take it that the movement that Mr. Buehler speaks about here, that may have been up and down, was probably much more likely a rotational phenomenon of some sort and that therefore the fracturing is more accentuated than it would be by merely an up and down movement, since these high ridges and knobs of granite or porphyry sticking up into the sediments make pivots or areas of more or less stability.

In almost any deformation by tension, the direct pull, of course, may produce fracturing along shear planes inclined to the direction of tension and differential movement along those planes. The rocks are very likely to break in a manner that produces fractures that everybody knows as "shearing." Also, tensional stress will produce breaking at right angles to the tension and hence produce certain fractures which are obviously purely tensional.

I should say that what Mr. Lyden was saying a while ago about the distinction between "shearing by tension" and "shearing by compression," in so far as it can be identified, was this: In the shearing by tension, the movement is taken up by a great series of small fractures which, instead of running continuously for considerable distances, as they would in a compressional deformation, are continually feathering out and curving erratically; and that where fractures are definitely identified as shearing due to compression they are more continuous and straighter, tend to align themselves more nearly parallel to each other and do not feather out in these little curlycues at the end. Is that right?

J. P. Lyden.—That is right.

G. M. Fowler.—That is right.

G. M. Fowler and J. P. Lyden (written discussion).—Answering Dr. Dake's observations regarding chertification, it is our belief that it is due in many instances to replacement of the limestone along shear zones instead of always being a surface phenomenon. The former is largely the condition throughout the Tri-State district and other parts of the Ozark region with which we are familiar. In the Tri-State district some chert may have been formed at the surface by the process described by Dr. Dake and Dr. Leighton, but the great bulk of it is due to the silicification of the limestone, often far below the surface that existed at that time. Chert, 40 ft. or more thick., frequently is found in drill holes and underground workings in the Picher-Miami area.

Zones of shearing are common throughout the Ozark uplift. Many of these zones are chertified. As drainage developed these shattered zones naturally became, in many instances, the stream channels, because they eroded more rapidly than the massive limestone. Shearing is in evidence in the bluffs bordering many of the streams. These bluffs are often chertified. Under such circumstances it is natural to find chertification, nearly or entirely absent, a short distance back from the face of the cliffs, just as Dr. Dake has described them. The same reasoning applies regarding the drill holes that he mentions. We have made many similar observations several hundred feet underground in the vicinity of Picher, and along the bluffs bordering many of the streams in this vicinity. Chert can be a surface phenomenon, but it can form also in great quantity many feet below the surface.

Dr. Buehler commented upon the fact that ore-bearing solutions from a magmatic source would have to pass through 1000 ft. or more of sedimentary formations in order to reach the horizons in which the orebodies of the Tri-State district are found. He asked why ore failed to deposit in these lower formations. We might also add that ore-bearing solutions from a meteoric source would have to pass, in many instances, through approximately 100 ft. of limestone or flint, as formations of this thickness overlying the ore horizons are nearly always barren. Also, the strata between the

highly mineralized ore beds are barren, except in areas of intense metamorphism. We believe that the strata mentioned failed to mineralize because they were of such a nature that favorable reservoirs for ore deposition did not form in them. Generally they were massive beds of flint, limestone or dolomite.

In nearly every mining region in sedimentary rocks with which we are familiar the ore deposits are confined largely to one or more favorable strata. Often the ore-bearing solutions, regardless of their source, passed through many feet of barren formations to reach the favorable horizons. Districts with orebodies of this character are Leadville and Redcliff, Colo.; the southeastern Missouri lead district; Goodsprings, Pioche and Unionville, Nev., and Metaline Falls, Washington.

In order to show the intensity of mineralization where conditions were particularly favorable, the production data of the Woodchuck mine may prove of interest. This mine comprises 40 acres in the Picher-Miami field—description as follows: SE.¼ of NE.¼, sec. 30, T. 29 N., R. 23 E.

The Woodchuck mine was opened in 1915, its aggregate production to Dec. 19, 1931 follows:

Aggregate rock tons of crude ore milled, 1,557,867

Aggregate zinc concentrates produced, 103,454.785 tons with metallic zinc content of 59.25. per cent.

Aggregate lead concentrates produced, 24,125.797 tons with metallic lead content of 82.60 per cent.

G. M. Fowler and J. P. Lyden (written discussion*).—A recent interesting paper by Dr. C. K. Leith[6] makes several references to our paper on the Ore Deposits of the Tri-State District.

On page 405, regarding the relation of structure to ore genesis Dr. Leith says: "In some cases the investigator has had such a strong predisposition toward one or another hypothesis of origin of the ore that he has obviously been influenced in the selection of his supporting structural facts from the many available."

On page 417: "Comparing these various views, it is apparent that advocates of deposition of the ore by rising waters, whether hot or cold, have desired to see evidence of deep fissures and faults controlling the runs of ore, and have not hesitated to assert definitely that such fractures exist, even though their observational evidence was of a highly circumstantial variety. Although other than structural evidence has been cited by advocates of hot rising solutions, such as assumed zonal arrangement of ores and the existence of scattered dikes in the same formations in distant areas, nevertheless the existence of deep faulting and fissuring has bulked large in their argument,—so much so that the impression has been created, particularly by Spurr, Emmons, and Fowler, that the acceptance of deep faulting carries with it a strong presumption in favor of rising hot solutions. To others there is an obvious non-sequitur in this argument."

On page 418, under Conclusions: "The desire to find confirmatory evidence of some hypothesis of ore genesis has clearly influenced the structural interpretations. Much of the structural evidence is therefore discredited as a sound basis for interpreting the origin of the ore. Other evidences of ore genesis are not here considered."

Evidently Dr. Leith overlooked our principal reasons for believing that the ores of the Tri-State district are from an igneous source, as stated in the two paragraphs at the bottom of page 238. This orderly and unrepeated sequence of the cycle of metamorphism and mineralization is, to us, the strongest evidence we have observed in favor of the ore from an igneous source.

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* Received Sept. 26, 1932.

[6] C. K. Leith: Structure of the Wisconsin and Tri-State Lead and Zinc Deposits. *Econ. Geol.* (1932) **27**, 405.

Fissures and fissure (shear) zones certainly do exist and are evident to anyone who will study this district in detail. We regard the fissures only as channelways and, in some instances, reservoirs for the mineralizing solutions. They are of equal importance whether the ore is from an igneous or from a superficial source. Dr. Leith infers that we mapped and accepted the fissuring in order to favor rising hot solutions. Careful reading of our paper should convince anyone otherwise.

We assumed it to be evident that our paper is concerned mainly with structure and its relation to the locus of the ore deposits. We have considered the origin of the ore merely incidental up to the present, and since it concerned us so little, our conclusions regarding it could hardly have influenced our observations on the major problem of ore finding.

On page 237, we say: "However, our observations in this district have impressed us with the fact that these orebodies and the structural conditions associated with them are similar in many ways to the orebodies in sedimentary rocks that are known to be in areas of igneous disturbance." This sentence is meant to refer to many relationships. Regardless of the source of the ore, whether leached from sedimentary formations or from deep-seated igneous rocks, our studies convince us that the delineation, size and shape, of all the ore deposits in the Tri-State district is due to structural features of the host formations. Additional observations regarding this point are given in the first paragraph under Structure on page 225.

Regarding our comparison of the structural relations of the Tri-State district and Butte, Mont., Dr. Leith says (p. 415): "Indeed, it is a clear case of extrapolation of a mechanical concept of Butte structure to a far distant and in many respects dissimilar field. Convinced of the correctness of their structural pattern and its origin, Fowler and Lyden assume that the fissures are deep-seated, and they favor the hypothesis of rising hot solutions from below, as in the case of Butte."

The fact of the matter is that we made the structural comparisons only after the study of thousands of recorded observations had directed our attention to the obvious similarity between the shearing in the Miami shear trough and that of the east-west age fissures in the Butte, Mont., district. The comparisons are more striking because the host rock here is sedimentary whereas that at Butte is igneous. At the beginning of our work we accepted the literature on the Tri-State field as a guide and commenced our studies with the notion that here was an entirely new structural problem. We were very much surprised, as others would be, to find ourselves dealing with problems the like of which we had studied elsewhere. To read into our conclusions the assumption that we have fitted the Tri-State problem into a set of preconceived ideas concerning it hardly seems justified.

On page 415 Dr. Leith says: "What these men describe as shear zones have been mapped by Siebenthal and Ageton as solution channels in which the Cherokee shales have been deposited, without positive evidence of faulting. The classification of fissures as shearing and tensional by Fowler and Lyden is clearly influenced by the requirements of the mechanical hypothesis which they have adopted. Others have not been able to make such a classification on the basis of observation of the fissures themselves."

Dr. Leith draws these conclusions from the work of Siebenthal and Ageton (pp. 412 and 413) in the Picher-Miami field; they mapped the shale-limestone contact, an old erosion surface which does contain many sink holes in the surface of the limestone. However, this old erosion surface does not reflect the true structure of the Boone strata beneath. In some cases troughs in the limestone as mapped by Siebenthal and Ageton are directly over structural domes in the Boone strata as mapped by us.

In the old Joplin field Siebenthal and Smith[7] recognized strata and correctly mapped structure, using the Short Creek oolite as datum. In most areas their work is excellent.

In many parts of the Tri-State district one may observe fractures and shattered areas that are definitely later than the ore deposits. They are largely of the gravity type. By doing detailed mapping they are readily distinguishable from the premineral fissures.

On page 225 we state: "In studying the structure of the Picher-Miami field we mapped orebodies, structural features and all ore-bearing horizons, both horizontally and vertically, as found in the mine workings and interpreted from drill-hole data This involved the correlation of the ore-bearing horizons over a number of square miles and under all types and degrees of alteration and deformation."

Our paper contains two maps and a vertical section (Figs. 12, 13 and 14) showing the Miami shear trough in the Angora-Blue Goose mine. These data were prepared from underground observations and drill-hole data in order to show actual conditions in the Miami shear trough. We have similar data in equal detail covering nearly 25 square miles in the important part of the Picher-Miami field. In fissure zones throughout the district vertical displacements of the strata may be measured readily, and in some areas exceed 100 feet.

Thousands of recorded observations determined the mechanical hypothesis for us; other influences were unnecessary.

On page 416 Dr. Leith discusses the Grand Falls chert and breccias and their relations to ore genesis in the Tri-State district: "Of especial interest is the varying interpretation of breccias in their bearing on ideas of ore genesis. These are widespread in the Tri-State field, and they constitute the host for much the greater part of the ore. In earlier interpretations these were regarded as mainly friction breccias, no matter what the form of their occurrence, and this idea is carried through to the present by advocates of rising hot solutions. From the start, minor parts of the brecciation were recognized by nearly all observers as resulting from solution and slump of limestone near weathered surfaces . . . When later, the Grand Falls chert breccia was found to extend in a great sheet down the dip into the Miami end of the field in a continuous layer 15 to 40 feet thick, roughly parallel to the bedding of the overlying and underlying sediments, it became difficult to account for it as a friction breccia, although this idea still persists among those who are trying to prove extensive faulting . . . I had occasion to call attention to it in connection with my study in 1925 of the silicification of old erosion surfaces . . . The evidence of the surficial origin of the greater part of the chert breccias now seems to me conclusive. Also I am entirely unable to conceive of structural deformation parallel to the bedding which could produce such results as the Grand Falls chert beds, even if the district had undergone far more deformation than is now apparent."

Chert is widespread in the Tri-State district. It is secondary in origin, having resulted from the silicification (chertification) of part of the Boone limestone. One of the horizons in the Boone has been designated by earlier geologists as the Grand Falls chert. In the Picher-Miami field this horizon is from 35 to 40 ft. thick and comprises beds O, P, and Q, described in our paper. It is a continuous stratum but it is not a continuous chert horizon. It is chertified and brecciated only in areas of structural disturbance and gradually grades from chert and chert breccia, or both, to limestone as the distance from the center of disturbance increases. The kind, size and shape of the breccia depend upon the degree and method of the disturbance and the character of the original rock.

Contiguous to disturbed areas the Grand Falls horizon sometimes consists of unbroken intercalated chert bands several acres in extent within the limestone.

[7] U. S. Geol. Survey Atlas, Joplin District Folio 148 (1907).

The positive identification of the beds of the Boone formation throughout the whole area is, of course, the key to any knowledge regarding the so-called Grand Falls chert. The significance of the detailed stratigraphy of the Boone is so great that it cannot be ignored by anyone presuming to understand the structure and alteration of the formation. There is nothing mysterious about the chert of this district, nor its relationship to the problems of ore finding.

The greater part of the chert in this district was formed beneath the old limestone surface—some near, some many hundred feet below. A small part of it may be of surficial origin. The areas of chert are discontinuous *laterally* and vertically and indicate numerous centers of disturbance. This is true throughout the entire Tri-State district. To account for the distribution of the chert as it actually occurs, it is entirely unnecessary to involve a great horizontal movement parallel to the bedding of the formation.

On pages 412 and 413, Dr. Leith discusses the relationship between the subshale surface and the ore trends. We have had the opportunity to study this supposed relationship in great detail and our conclusions regarding it are stated on page 232 in the paragraph beginning "In this district numerous attempts have been made . . . "

To those who are unfamiliar with this district, it may be of interest, and perhaps it is pertinent to add, that our knowledge of the district, as briefly summarized in the paper, is not the result of a few short visits or of the study of the work of other observers, but was gained after six years of continuous observation and study during which we were aided by two or more office assistants and by a number of engineers, employed by the several mining companies, who made surveys and collected many hundred samples of drill-hole cuttings for us to study. During the six-year period we have studied in detail the underground geology in more than 95 per cent of the mines in the Picher-Miami area. In doing this work we mapped the fissures, identified beds, noted rock alteration and mineralization, and established datum planes in the strata upon which to measure flexing and faulting of the beds. We critically examined and utilized these data, together with the logs of more than 18,000 churn-drill holes in this field. We compiled all these observations on maps of uniform scale for the purpose of visualizing the geologic relations of all or any part of the Picher-Miami field. Also, we studied in less detail many other areas in the Ozark uplift.

In all cases, the evidence and conclusions presented in this paper are based upon our actual observations. We believe that others, having made the same extensive observations and given the same amount of study to the problem, would draw similar conclusions. The recorded facts are ample to speak for themselves without the necessity of visionary speculation.

Nickel Resources, Production and Utilization

By E. S. Moore,* Toronto, Ont.

(New York Meeting, February, 1932)

Although nickel was in use in alloys long before the Christian era, the metal was not discovered until 1751, when Cronstedt recognized it in niccolite from Sweden. The Chinese apparently used a nickel alloy, known as "pahfong" thousands of years ago and Persian coins dating farther back than 200 B.C. consist of a copper-nickel alloy similar to some of those now in use in coins. The pure metal was first prepared about 1775 and the metal was refined for commercial purposes some years later.

The nickel industry did not attain much prominence until the ores in New Caledonia were discovered. Exports of ore from that country began in 1875 and since that date there has been an ever-increasing world output of nickel and a great expansion in the uses of the metal except during short periods of commercial and industrial depression. Previous to 1875, for a number of years, Norway was the chief producer, although the metal was mined also in Sweden, Finland, Italy, Germany, Austria, France, China, the United States and other countries. With the discovery of nickel deposits in the Sudbury district, Ontario, a new factor was introduced into the nickel industry and Sudbury has since become the overwhelmingly dominant feature in the world's nickel markets.

Although nickel was first discovered in Canada as early as 1848 near the north shore of Lake Huron and nickel-copper sulfides were found in 1856 by surveyors working close to the present location of the important mine at Creighton, in the Sudbury field, it was not until 1883 that any importance was attached to this field. In that year copper sulfide attracted attention where the main line of the Canadian Pacific railroad was being projected across Canada to the Pacific coast. Mining began near this discovery in 1889 but in the meantime operations had begun in 1886 in other parts of the field and by 1890 about 100,000 tons of ore had been raised. In the Sudbury field, as in Norway and at the old Lancaster Gap mine in Pennsylvania, the ore was first mined for its copper content and difficulty in the refining of the copper led to the discovery of the important proportions of nickel in the ore. No. 3 property of the International Nickel Co., now known as the Frood, the greatest deposit in the

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field and one of the most valuable single ore deposits ever found, was discovered in 1884 but relatively little attention was paid to it until recent years because the ore near the surface is low grade compared with that of many of the other deposits. New Caledonia continued to lead in the world's production until about 1905, when Sudbury forged ahead and has since held first place without a rival among producers. This field now supplies over 90 per cent of the world's nickel and in 1930 the output amounted to more than 120,000,000 lb. valued at nearly 24½ million dollars. The copper from this field in the same year had a value of almost 15 million dollars.

Distribution of Nickel Deposits

In addition to Canada and New Caledonia, the following countries, in order of rank, are producing small quantities of nickel from domestic ores: Norway, United States, Australia, India and Madagascar. France, Germany, Austria, Greece, Sweden, Finland, Italy, Turkey, Spain, Egypt, Great Britain, China, Mexico and South America have at some time produced small quantities of the metal, generally as a by-product from the smelting of other ores, as has been the case in the United States for many years. Other countries known to contain deposits, which are either too small to be of commercial importance or which have not been developed to a productive stage, are Brazil, Russia, Czechoslovakia, the Union of South Africa, Borneo and the Philippine Islands. So far as known Russia appears to have the most promising deposits controlled by these countries, although no production has been reported during recent years.

Canada

With the exception of a small quantity of nickel derived from the treatment of silver-cobalt ores from the Cobalt field in Ontario, the Sudbury field is the only one in Canada at present reporting a production of nickel. A number of nickel deposits are known to exist in different parts of the country and at least one, the Alexo, has produced some ore.

During the development of the Sudbury field many companies have played a role but there are at present only two producers, the International Nickel Co. of Canada, Ltd., and the Falconbridge Nickel Mines, Ltd. The former company, resulting from the amalgamation, in 1928, of the International Nickel Co. and the Mond Nickel Co., an English concern, controls the major portion of the nickel-copper deposits. It owns very large milling and metallurgical plants, including a concentrator and smelter at Copper Cliff, a smaller smelter at Coniston, Ont., nickel refineries at Port Colborne, Ont., and Clydach, Wales, and a platinum metals refinery at Acton, England. This refinery has a capacity of 300,000 oz. of platinum metals per annum. There is also a large copper and precious metals refinery at Copper Cliff, Ont., in which this company

holds an important interest. It has a capacity of 10,000 tons of electrolytic copper per month. Since the amalgamation, this company has spent nearly 50 million dollars in the development of its mining and metallurgical plants, so that it is now in a position to produce nickel and copper on a very large scale and in the most efficient manner. In connection with the smelters of this company, Canadian Industries, Ltd., operates chemical works for the manufacture of sulfuric acid and niter cake (using sodium sulfate from Saskatchewan) at Copper Cliff.

If running at full capacity the existing mines of the International Nickel Co. could hoist monthly about 380,000 tons of nickel-copper ore and produce 300,000 tons of sorted ore. The Frood mine alone, although now operating on a curtailed basis, has a daily capacity of 8000 tons of ore. The ore reserves of this company were reported in 1930 to be 206,704,000 short tons (2000 lb.) distributed among the properties approximately as follows:

Mine	Tons	
Frood	136,000,000	91,000,000 tons low grade; 45,000,000 tons high grade. Low grade carries about 3.75 per cent Ni + Cu; high grade over 5 per cent.
Murray	22,490,000	
Levack	19,000,000	
Stobie	13,700,000	
Creighton	7,289,000	
Garson	4,324,000	
Cream Hill	3,028,000	
Other properties	959,000	

In 1930 the ore smelted from the Sudbury field amounted to 2,359,154 short tons. The nickel content of the matte produced was 61,112 tons and copper 70,800 tons, giving an average content of 2.6 per cent nickel and 3.0 per cent copper. The Creighton ore is richer in nickel than most of the ore from the other deposits and is now used almost entirely for the manufacture of monel metal. The lower portion of the Frood orebody is very rich in copper and from the 2800-ft. to the 3100-ft. level the content runs from 12 per cent to over 20 per cent of this metal. The copper ore high in copper is rich in platinum metals and gold and silver. It is reported that ore from workings on several of the levels from 1600 to 2800 ft. carries $50 in precious metals per ton of bessemer matte.

Falconbridge Nickel Mines, Ltd., has one mine operating in the Sudbury field. The ore reserves already blocked out on this property are estimated by the company at about 2,725,400 tons, averaging 2.31 per cent Ni and 0.94 per cent Cu. A 300-ton smelter is situated near the mine and the matte is shipped to Kristiansand South, Norway, where the Falconbridge Nikkelverk operates a refinery with a capacity of about 2500 tons refined nickel per annum. The matte treated in this refinery is drawn from Ontario and Norway in the proportions of about 70 per cent from Ontario and 30 per cent from Norway.

While the figures quoted above as estimates of the tonnage of ore reserves for the Sudbury field cover the deposits so far drilled or developed, it is only reasonable to assume that very large bodies of ore still remain undiscovered. This assumption is based on the discovery within comparatively recent years of large reserves at the Frood, Levack, Murray and Creighton mines, and considerable deposits on other properties. Several very valuable orebodies have been found at the Creighton since it was believed that the property was approaching exhaustion. In 1930 the International Nickel Co. of Canada increased the ore reserves of its working mines 6,610,000 tons, a sufficient quantity of ore to furnish nickel for three years at the 1929 peak production rate.

All of the nickel produced from the Sudbury district comes from the norite around the Sudbury nickel eruptive or from offshoots from this intrusion. There are, however, in the counties lying southwest of Sudbury, several small pyrrhotite-chalcopyrite deposits carrying low values in nickel and copper. These occur in the Nipissing diabase, which is believed to be of the same age as the nickel eruptive of Sudbury.

The Alexo mines, situated in Dundonald township, Ontario, 150 miles north of Sudbury, shipped ore to the Coniston smelter of the Mond Nickel Co. for several years following 1912. The ore occurs with serpentine and peridotite and as massive and disseminated sulfides as in the Sudbury field. It averages a little higher in nickel than the Sudbury ore, the average of the ore shipped being about 4.5 per cent Ni. The ore is said to be practically self-fluxing, owing to high magnesia content. Some thousands of tons of ore have been shipped and there is apparently considerable ore remaining in the deposit but no estimate of reserves is available to the writer.

Nickel ore has been found on Lake Shebandowan in western Ontario, and in eastern Manitoba on the Oiseau and Maskwa rivers, but while the ore is of good grade no large bodies have been developed. Near St. Stephen in New Brunswick there are deposits of nickel in basic rocks but so far any attempts to develop them on a commercial scale have been unsuccessful. They carry less than 2 per cent Ni and 1 per cent Cu. One estimate places the tonnage of one lens at 150,000 tons.

Considerable interest has been aroused by the discovery of nickel on Ranken inlet on the west coast of Hudson Bay. In a description of this deposit Drybrough[1] gives the following estimate of ore reserves derived from drilling operations:

	Tons	Cu, Per Cent	Ni, Per Cent	Pt, Oz.
Massive sulfide	30,000	1.28	9.25	0.20
Disseminated sulfide	90,000	1.19	3.10	0.08

[1] J. Drybrough: Nickel-Copper Deposit on Hudson Bay. *Bull.* Can. Inst. Min. and Met. (1931) 390.

Milling tests indicate that the low-grade ore can be concentrated satisfactorily. The ore consists chiefly of pyrrhotite and chalcopyrite in basic rock and is much like that at Sudbury. Another deposit of lower grade has been found in the same area.

New Caledonia

Nickel was discovered in this French possession in the South Pacific ocean in 1865, by M. Jules Garnier, and the first ore was shipped in 1875. This island rapidly became the main source of the world's nickel and held first place as a producer until 1905, when surpassed by Sudbury. The maximum annual production—about 133,000 metric tons of ore—apparently was reached about the beginning of the present century and it then declined slowly for a number of years until in 1930 the production of the metal was only about one-half what it was at the time of maximum production. This falling off in output is partly due to the decrease in the metallic content of the ore mined. The ore has been shipped mostly to Great Britain, Belgium, France and Germany for metallurgical treatment. High costs of shipping ore such great distances have always militated against the successful operation of the mines and smelting on the island has had a checkered history. Smelting was begun in 1880 and shipments of both ore and matte continued for five years. Smelting ceased for three years and was renewed on a small scale for three years more, then abandoned to be renewed once more in 1910. Shipments of ore have decreased and for several years apparently only matte has been shipped from the island.

The New Caledonia ore consists of garnierite or noumeite, a hydrous nickel-magnesium silicate, free from copper but mixed with considerable iron and in places associated with some cobalt and chromium. The deposits are surficial in nature and occur in shallow basins. The mines are surface workings. The ores are much higher in nickel than those of the Sudbury field, the richer types formerly mined carrying 6 to 10 per cent and those mined in recent years probably 4 to 6 per cent of the metal. They have also had an advantage, not so evident now as in the earlier years of the industry, in being more easily treated metallurgically than the complex copper-nickel sulfide ores, but the deposits are relatively small. The largest deposit mined is said to have produced 600,000 tons of ore while an orebody is rare which produces over 250,000 tons. No satisfactory estimate of ore reserves is available since it is so difficult to make an estimate of the volume of bodies of such irregular outline. In 1917 the late W. G. Miller made a rough estimate of 160,000 tons of nickel as probably the quantity of metal that might be derived from these deposits. This quantity is approximately equal to that produced by the mines between 1875, when mining began, and 1917.

United States

The United States has been a small producer of nickel ore for about 160 years but, except for a few years before nickel was first mined in New Caledonia and when the old Gap mine in Pennsylvania was producing considerable amounts of the metal, she has not been able to supply more than a very small fraction of her very large requirements. Nickel deposits of a large variety of types are scattered widely over the country and it is remarkable that no important deposits have been discovered. According to *Mineral Industry*, only 305 long tons of the metal was produced in 1929. In 1927 production was 728 long tons and this greatly exceeded the production for many years previous to that date. The metal is derived entirely from the smelting of ores of other metals.

For a number of years the lead ores of Mine La Motte in Missouri furnished considerable nickel and cobalt as by-products but this source has been practically exhausted.

The Cuban iron ores, which are largely consumed in the United States, contain a small quantity of nickel and it has been utilized in the manufacture of steel rails, thus strengthening the steel without the necessity of extracting the nickel from the iron.

South America and Mexico

Nickel has been recovered with cobalt as a by-product from ores from several localities in Mexico and South America but records of the production of the different countries are not available. The quantity has been small and no typical nickel deposits have been found except possibly in Minas Geraes, a province of Brazil where surficial deposits of garnierite have been reported to exist.

Europe

Several countries in Europe have produced small quantities of nickel. The most important and consistent producer has been Norway, which for a time before the discovery of the New Caledonia ores was the world's most important source of this metal. Mining began before 1850 and production reached a maximum in 1876, when it is recorded that 360 tons of the metal were derived from 42,500 tons of ore. The ore was low grade, carrying only about 1 to 1.5 per cent Ni. The price of the metal, however, was much higher then than it has been in recent years.

The opening of the New Caledonia mines caused a falling off of nickel mining in Norway and most of the mines there and also some small ones in Sweden and Finland were permanently closed. There has been a revival of mining in the last few years in Norway and over 400 long tons of the metal was produced in each of the years 1928 and 1929. The opening of the refinery of the Falconbridge Nikkelverk at Kristiansand

South has been responsible for stimulating mining, as about 30 per cent of the matte treated at this plant is drawn from local sources on a toll basis. The ores are of the nickel-copper sulfide type associated with norite and other basic rocks and are regarded by Vogt as magmatic segregations of these basic rocks similar to the ores of Sudbury.

Russia contains a number of nickel deposits in the Urals and some of them seem to be of considerable promise. Apparently they are of surficial type. Attempts have been made at various times since 1866 to mine them, and while a small output has been mentioned at several periods no estimates of resources are available, nor has any production been recorded in recent years.

Great Britain, France and Germany have been important refiners of nickel. Much New Caledonia ore has been shipped to these countries for treatment. They have all possessed local deposits containing nickel. In Great Britain, pentlandite has been mined on a small scale and some copper, iron and tin ores carry the metal. In France small quantities of nickel and cobalt were found in the eighteenth century in the silver ores at Chalanches. Nickel and cobalt have also been derived from the smelting of other ores in Germany and Austria but no regular deposits of nickel ore are recorded in these countries. Considerable nickel was found in the ores of Schneeberg, Saxony.

Greece has been known for many years to contain nickel, and small quantities of ore have been shipped to Norway and other European countries for treatment. The metal occurs as silicate and in nickeliferous iron ore. A few tons of the metal have been produced within the past 10 years but the production is small and desultory.

Czechoslovakia, Turkey and Spain all contain deposits containing a little nickel but no production has been reported from these countries in recent years, although the metal was mined in Spain during the last century. Italy has produced a little of this metal from typical nickel deposits of small size but geologically similar to those at Sudbury.

Africa

A little nickel ore of good grade has been shipped from Egypt but no important deposits are known to have been found. In Cape Colony, near Insizwa, deposits of nickel with basic rocks have been found but so far they have not been proved to be of economic importance. The Vlakfontein nickel-copper deposits of the Rustenburg area, Transvaal, occur in norite of the Bushveld complex and have been described by Wagner[2] as magmatic segregations similar to those at Sudbury. These ores vary from disseminated sulfides to massive sulfides. There are several pipelike orebodies, some of them separated by 1000 ft. or more

[2] P. A. Wagner: Magmatic Nickel Deposits of the Bushveld Complex. Dept. of Mines, Pretoria, Transvaal, Union of South Africa, *Mem.* 21 (1924).

from each other and small lenses are strung along a zone more than 15 miles long. Considerable development work has been done but none of the orebodies discovered so far are very large. While no production has been recorded, it seems possible that this area may produce commercial nickel ore, as some of the geological conditions strongly resemble those of the Sudbury field.

Madagascar has been known for 15 years or more to possess nickel silicate deposits similar to those in New Caledonia but the production up to the present has been very small.

Asia

China has employed nickel in alloys used in coins and other articles since very early times but her production has not been important. It has come chiefly from the smelting of ores of other metals from southern China, including some nickel-copper ores.

India is the most important producer in Asia. Nickeliferous pyrrhotite is mined with the copper ores of Rajputana and nickel is found with the ores of gold and other metals at several places. The production in 1929, according to *Mineral Industry*, amounted to 830 long tons of speiss.

Australia

Ores carrying nickel and cobalt have been found at a number of places in Australia but Tasmania is apparently the only section of the country recording any appreciable production of nickel. The deposits on this island consist of nickel-copper sulfides carrying a little silver and platinum. They are along the borders of masses of serpentine and gabbro and while the ore is of excellent grade, much higher in nickel than most of the Sudbury ore, the deposits so far discovered are small. The production has been of a desultory nature. In 1929 the output was about 95 short tons of metal.

East Indies and Philippine Islands

Borneo and the Philippine Islands contain surficial iron ore deposits carrying nickel and chromium, very similar to the iron deposits of Cuba. No record has been found of nickel production from either of these areas.

Uses of Nickel

Nickel is believed to have been first used by man as an alloy of iron obtained from meteorites. Later it was employed by the Chinese in alloys with copper or silver without any definite knowledge of its presence in the alloy, since it was not separated from the other metals by refining processes. It continued to be so employed by the Chinese and other peoples for centuries and the uses of these alloys were extended slowly

for other purposes. Electroplating and the rolling of malleable nickel on iron and steel plates made new demands on the market for the metal. Although Faraday had alloyed nickel and iron in 1820, nickel-iron alloys were not patented until 1870. Further, it was not until Marbeau in France produced ferronickel and nickel steel in 1885 that an extensive use for nickel was developed. In the meantime the large deposits of the metal in New Caledonia and Ontario had been discovered and there had been a considerable output of ore from the former country. The lack of markets caused a great fall in price of the metal. In 1873 the price had risen to over \$3.80 per pound but it fell to \$0.90 in 1879 and \$0.16, the lowest ever recorded, in 1886. Since that date the price has never exceeded \$0.75 per pound nor fallen below \$0.26, in spite of the greatly increased demands for the metal. These have been met by the enormously augmented production of ore.

The consumption of nickel in the manufacture of armor plate and other types of ordnance became the main factor in the markets and continued to be so until the end of the war, when a great slump in the demand for the metal occurred. The industry which had been built up to a high peak of production was faced with a collapse and it was only the courage of the men faced by this dilemma that made recovery possible. The companies engaged in the industry, particularly the International Nickel Co., inaugurated an extensive program of research into the uses of the metal and in a few years a great variety of new uses resulted in a bigger, more stable and varied market than had formerly been available. In the years preceding the present depression the production and consumption far exceeded that of the former peak years of the war.

The uses of nickel as the metal, or in alloys and salts, are now legion and it is impossible in a paper of this sort to mention all the machines, articles and processes in which the metal is employed in some form. Following, however, is an outline of forms in which it is used and the approximate percentages of the nickel production that goes into these different materials. The writer is indebted to the International Nickel Co. for this information.

Nickel Steels.—Nickel, nickel-chromium and nickel-molybdenum (0.5 to 5 per cent Ni). These steels consume about 32 per cent of the nickel and they are used extensively in motor cars, trucks, tractors, locomotives, mining and mill machinery, bridges, shovels, heavy guns, armor plate and a vast number of other machines and structures.

Corrosion-resistant Steels (7 to 35 per cent Ni).—About 5 per cent of the production is consumed in these steels and they are used in the manufacture of cooking utensils, marine fittings, turbines, chemical apparatus and many other articles. Stainless steel, which consists on the average of 8 per cent Ni, 18 per cent Cr and balance iron, is an alloy of this type which gives promise of creating an ever-increasing demand for nickel.

Ferronickel Alloys.—These consume about 4 per cent of the output. They may be divided into three groups: (1) Low-expansion alloys (35 to 45 per cent Ni), which are used chiefly in automobiles, engine pistons, exact dimension rods, and tapes, thermostatic metals and electric light bulbs; (2) magnetic materials (45 to 80 per cent Ni) used in submarine cable sheathing, radio transformers, telephone and telegraph relay parts and special type current transformers; (3) nonmagnetic materials (10 to 25 per cent Ni). Used extensively in electrical construction.

Nickel-steel Castings (1.5 to 4 per cent Ni).—For manufacture of locomotive frames, crushers and crusher jaws, tube balls, high-pressure valves, etc. About 2 per cent of the metal goes into the manufacture of such castings.

Nickel Cast Iron (1.5 to 3 per cent Ni).—There has been a considerable development in the use of nickel cast iron in the metal industries in recent years and this consumes over 5 per cent of the output of nickel. It is employed extensively in automobile cylinders and pistons and in steam engine and pump cylinders, pistons and rings. Also in oil engines, printing machinery, dies, machine tool tables, benches, etc. Special types of nickel cast iron with much higher nickel content are used for subway and surface car resistance control grids and car heating grids (5 per cent Ni) and for electrical machinery castings (10 to 15 per cent Ni).

Nickel-silver (10 to 30 per cent Ni).—This is a copper-nickel-zinc alloy used in plated ware, hardware, plumbing and so forth.

Copper-nickel Alloys (15 to 50 per cent Ni).—Employed in manufacture of bullet jackets, corrosion-resisting castings, valves and valve trim.

Nickel for Coinage (25 to 100 per cent Ni).—The consumption of the metal for coinage, copper-nickel alloys and nickel-silver takes about 7 per cent of the total production.

Heat-resisting Alloys (35 to 80 per cent Ni) *and Electrical Alloys* (25 to 80 per cent Ni) consume about 8 per cent of the output of nickel. The former are used in electric-furnace heating elements, structural furnace castings, tubes and retorts in the chemical industry and the latter in rheostat and pyrometer wire.

Nickel Anodes and Nickel Salts.—Used in nickel plating. These account for about 8 per cent of the nickel produced.

Nickel Bronzes (0.5 to 5 per cent Ni).—About 0.5 per cent of the metal goes into such bronzes, which are used in bearings, valve castings, steam packing metal and pressure castings.

Rolled Nickel (pure, malleable, 99 per cent Ni).—Rolled nickel is used in dairy equipment, hotel kitchens, cooking utensils, food-canning establishments, chemical equipment, laboratory apparatus and for many other purposes, and about 8 per cent of the metal is employed in this way.

Miscellaneous Uses.—Aluminum and zinc-base die castings (0.5 to 5 per cent Ni) and white gold (6 to 25 per cent Ni). Nickel as a catalyzer

for hydrogenation of edible oils and nickel elements in storage batteries. These miscellaneous uses consume about 1.5 per cent of the metal output.

Monel Metal (Ni, 68 to 72 per cent; Cu, 28 to 32 per cent; iron about 1.5 per cent and varying proportions of manganese). This is a patented alloy discovered in 1905 and since that time manufactured and marketed by the International Nickel Co. of Canada. The ore from the Creighton mine is practically all used in the manufacture of this alloy because the proportions of nickel and copper that it contains make it easy to manufacture the alloy from matte made from this ore. There has been a great expansion in the use of this alloy and about 19 per cent of the output of the metal is now absorbed by it. The alloy is noncorroding, strong, tough, of high tensile strength, and it can be rolled. It is extensively employed in airplanes, automobiles, doors and other parts of buildings, chemical laboratories, kitchens, hospitals, laundries, soda fountains, railroad cars and ships and in other ways too numerous to mention.

It is evident from this outline that the past 10 years have seen a tremendous expansion in the use of this metal. There are also many indications of a continued increase in uses to which it may be put. Many alloys now in use contain at least small proportions of the metal and the number of these is constantly being increased.

Production and Prices

Mineral Industry gives the following figures for the production of nickel in 1929 from the following countries in long tons (2240 lb.): Norway, 431; Tasmania, 85; United States, 306; India, 830 (nickel speiss).

In 1930 the Canadian production of bessemer matte was 166,703 tons (2000 lb.) and its nickel content 61,112 tons. A small quantity of nickel —58 tons—was recovered from silver-cobalt ore mined at Cobalt, Ont. The production of New Caledonia for the same year is reported as 6743 metric tons of matte, which may be assumed to carry about 73 per cent Ni.

The world's production of nickel since 1850 in 10-year periods, together with the Canadian production since 1890 in 5-year periods is approximately as shown in Table 1. These statistics indicate the continuous growth of the nickel industry over a period of 80 years and also the dominating position of the Canadian producers in the market.

The production of nickel in 1931 will doubtless show considerable decrease from that of 1930, as some of the mines of the International Nickel Co. of Canada are operating at 25 to 35 per cent capacity and the Levack mine is temporarily closed. Sales of nickel in 1930 were much below those of 1929. There is one favorable feature in the situation from the standpoint of the mining man—that the company is not exceeding the stocks of the metal which it was estimated, at an early stage in the

TABLE 1.—*Production of Nickel*

World Production		Canadian Production	
Year	Short Tons	Year	Short Tons
1850	25	1890	1,750?
1860	180	1895	2,315
1870	445	1900	3,540
1880	625	1905	9,503
1890	4,100	1910	18,636
1900	10,000	1915	34,045
1910	24,500	(1918)	(46,072)
(1918)	(47,300)	1920	30,568
1920	32,500	1925	36,596
1930	66,000[a]	1930	61,170

[a] Estimated.

depression, it might be safe to carry. Further, these stocks are much smaller in proportion to the conditions of the market than they were during the industrial collapse in 1921.

The price of nickel for many years has fluctuated much less than that of many of the other metals. It has been very stable since 1887 compared with prices previous to that year. In 1840 it was as high as $1.70 per pound; in 1845–46, $3.05. By 1862 it was down to $1.10 but it rose in 1873 to the highest point recorded, $3.80. By 1886 it was down to $0.16, the lowest on record. The price recovered and by 1892 reached $0.75, to decline to $0.30 in 1895. Between 1902 and 1918 the price remained nearly uniform between $0.40 and $0.45, except in 1917–18 when it rose to $0.50. At this time there was about $0.10 difference in the price of shot and electrolytic metal but as all the metal is now refined electrolytically such differences do not exist. Since the close of the war the price has fluctuated between $0.40 and $0.28 but it has been practically fixed during the last five years at $0.35 per pound. The prices quoted above represent the extremes of those on record. The International Nickel Co. of Canada, which controls the major portion of the production, has for some years made it a policy to maintain practically a fixed price during periods of inflation such as that experienced in 1928 and 1929 and depression such as that through which we are now passing.

ACKNOWLEDGMENTS

The writer wishes to thank Mr. C. E. MacDonald of the International Nickel Co. of Canada for information relating to the affairs of that company and for his kindness in reading over this paper. He also acknowledges his indebtedness to several authors for valuable informa-

tion gleaned from their works. These include the various authors of the Report of the Ontario Nickel Commission, Prof. A. P. Coleman, author of The Nickel Industry, published by the Mines Branch, Ottawa, Canada, and Mr. Robert C. Stanley, author of Nickel, Past and Present, *Trans. Can. Inst. of Min. and Met.* (1927). These excellent papers contain a wealth of information relating to nickel.

Abstracts

On the following pages are abstracts of papers published by the Institute during the year 1932 as Technical Publications, Preprints, in bound volumes and in Mining and Metallurgy. For abstracts of papers that appear in bound volumes in 1932 but that were published as Technical Publications in 1930 or 1931, see the Transactions for those years. Papers that appear in this volume are not abstracted here.

Many of the Technical Publications have been reprinted in bound volumes. Information regarding this disposition, and number of pages in each paper, may be found in the list beginning on page 309.

Volume numbers are given as listed on page 3 and in the paragraph at the beginning of the index, page 315. The abstracts are grouped as follows:

Metal Mining

The Mineral Industry. By Scott Turner. (*Min. & Met.*, June, 265. 3500 *words.*) Abstract of an address delivered before the Royal Canadian Institute, Toronto, April 16, briefly outlining the importance of mining in the United States, in Canada, and in the world. Comparison is made of the mineral production for 1886, 1900, 1910, 1920 and 1929, showing that in the 43 years from 1886 to 1929 that of the world increased nearly tenfold, of the United States over twelvefold and of Canada thirtyfold, but that Canada's output was equivalent to only 2 per cent of the world total and to 5 per cent of that of the United States in 1929. Statistical tables and graphs show value of production, percentage of world production contributed by Canada and the United States, etc. The mining situation in Canada is briefly reviewed: only those American engineers who have had occasion to examine the figures appreciate the rapidity of growth of Canadian mineral production. The treatment of minerals involves vast complex industries. When we are producing, say, 6 billion annually, the employment of 2 million workers provides indirect support to perhaps 10 million people. Exploiting mineral deposits is peculiar in certain ways, for the deposits are exhaustible, are of rare occurrence when measured in terms of outcrop area compared to total earth area, and may suddenly be handicapped or superseded by new discoveries; moreover, the economic position of the contained metals is affected by the use of scrap metal and the existence of invisible stocks. The largest and most valuable of the mineral deposits are not the metals, but rather the fuels. The record of the coal industry in recent years is one of brilliant technical achievement; its economic position is far from satisfactory. A so-called world fuel

surplus exists and fuel economy has become an organized movement. Overproduction and falling prices characterize the present position of the petroleum industry. More recently there has been a world tendency to overproduction in the metals. "It seems pertinent to suggest that the warning signals be heeded with respect to the metals."

Progress in the Improvement of Methods and Equipment at Open-pit Iron Mines on the Lake Superior Iron Ranges. By MAX H. BARBER. (*Tech. Pub. No.* 487. 3000 *words.*) Progress has been one of evolution rather than new development, with the exception of the application of electric power to replace steam. Improved practice shows a tendency to blast large areas in order to break ore or rock for seasonal operations, and unit cost of blasting has been materially reduced. Modern drilling equipment has developed into heavy, all-steel, self-propelling units, capable of drilling deeper holes at a much faster rate. These machines are also being used in structure drilling for exploration. The outstanding improvement in shovel equipment has been the development of the full-revolving electric shovel, mounted on caterpillars. Tendency has been toward the 4 to 5-yd. electric for ore, and the 8 to 10-yd. machines for stripping. Coal strip shovels have been constructed up to 18 cu. yd. capacity. Caterpillar traction has effected a big saving in labor and the maintenance cost of the electric shovels is much lower than steam equipment. Future development is expected to bring out lighter, more powerful shovels, through use of alloy steels. There is a noticeable increase in the use of small shovels for clean-up in shallow digging. The 16 and 20-cu. yd. air-dump cars are being replaced by 30-yd. equipment. Electric haulage is gradually coming into more general use. Diesel engines have been tried on the Mesabi Range, but without much success. Beneficiation of open-pit ores is receiving attention and the tonnage of treatable ore in the open-cut mines of the Lake Superior district will probably be materially increased as the result of improved processes.

Technical Advance on the Mesabi Iron Range. *See* MILLING AND CONCENTRATION.

Intermittent Mine Ventilation, an Economy Measure for Periods of Reduced or Suspended Operations. BY OSCAR A. GLAESER. (*Min. & Met.*, April, 169. 4300 *words.*) Describes a method of ventilation worked out by the author at the United Verde copper mine, at Jerome, Ariz., which effected important economies and improved conditions during either curtailed or suspended operations.

Current Position of the Copper Industry. BY L. VOGELSTEIN. (*Min. & Met.*, December, 527. 1100 *words.*) The reimposition by the United States of an import duty on copper and the duty recommended for the British Empire at the Ottawa Conference will bring about a decided change in the marketing of copper. Improvement in general business, especially the electrical business, and a sane policy by producers, are needed to restore the world's copper industry to a profitable basis.

Renewed Activity in California Gold Mining. By WALTER W. BRADLEY. (*Min. & Met.*, September, 385. 6500 *words.*) At one time or another gold has been mined to some extent in at least 40 of California's 58 counties. The total recorded yield to the end of 1931 is $1,852,470,376. Production exceeded $41,000,000 annually from 1850 to 1861 inclusive, with 1852 the maximum year at $81,294,700. From 1900 to and including 1917, production exceeded $16,000,000 annually, with a maximum of $22,442,296 for 1915, due to activity in both dredging and the deep lode mines. With output of other minerals and metals curtailed, owing to overproduction with consequent low prices and to other adverse economic conditions, the position of gold is relatively better today than it has been at any time since the World War upset the economic balance of industry and finance. The "Gold Rush of 1932" has sent thousands of men (and some women and children) into the hills, seeking the yellow metal on every stream and gulch and cañon, utilizing every conceivable sort of con-

trivance. The average return appeared to be possibly 40 or 50¢ per day. The interesting item of note in California gold mining is the increasing number of flotation plants at work and being installed; also an innovation in "crushing" devices—the Hadsel mill. Inquiries to the State Division of Mines for gold properties, mainly for lode, but occasionally for hydraulic and drift, are especially active. From many sources come reports that examinations are being made, engineers are scouting for properties, options are being given, and mines are being reopened.

Gold Lodes of the Willow Creek District, Alaska. By JAMES C. RAY. (*Min. & Met.*, September, 409. 4500 *words.*) The Willow Creek district, in the southwestern portion of the Talkeetna Mountains, is reached by a good automobile road from Wasilla, a small town on the Alaska Railroad about 45 miles north of Anchorage. The gold veins, which belong to the mesothermal type, occur in a quartz-diorite of batholithic character. Gold of unusual purity occurs with quartz, arsenopyrite, pyrite, tetrahedrite and galena, and minor amounts of sphalerite and chalcopyrite. The gold was the last metallic mineral to be formed. It occurs interstitially in the quartz and as replacements of the other metallic minerals. The principal zone of mineralization extends across the district for about 8 miles in a northeasterly-southwesterly direction. This is also the general direction of the strike of the veins which dip to the northwest. Numerous post-mineral transverse faults cut all the known veins. The longest vein segments in developed properties are about 1200 ft. In the southwestern portion of the district the average value of ore mined has been about $50 per ton. The gold content of the ores appears to diminish towards the northeast to $25 per ton. The fine grinding necessary to free the fine gold particles is best accomplished in the ball mill. At the Lucky Shot mill 85 per cent of the gold is recovered in the top cut from a Gibson table, the sulfide concentrates from the table and flotation cells pass through an amalgamation pan in close circuit with a small Harding ball mill. All siliceous tailings are rejected and final treatment of the sulfides consists of cyanidation. In 1931 the total production of the district had been considerably over $5,000,000. A single oreshoot has produced more than $600,000 in gold. The character of the vein mineralization and rock alteration are similar to that at Grass Valley and Nevada City, Calif., and it is the writer's opinion that the veins will continue to a depth of several thousand feet without material change in the character of ore and its gold content.

The Outlook for Silver: Present and Future. By C. W. HANDY. (*Min. & Met.*, December, 519. 1800 *words.*) Supplies of newly mined metal have been greatly curtailed and supplies from melted coinage promise not to be so great as in the past, but both of these should be favorable market factors. The demand for silver for coinage is improving; for use in arts and industries is declining, but not seriously. The shrinkage in demand from India and China is the chief cause of low silver prices. The trade of those countries has been crushed by the fall in price of their exports and by the impaired purchasing power of their foreign markets. With an improvement in these conditions will come higher prices for silver.

Gold: Its Production and Marketing. By F. W. BRADLEY. (*Min. & Met.*, September, 401. 4500 *words.*) A record of the present situation in the production and distribution of gold, with discussion and interpretation by the author, who points out that though gold is found in every formation constituting the earth's crust, though it is possible to find it free in great concentrated placer deposits on the surface, though there are still large unprospected areas of both the surface and the crust, though it may be possible to produce it by the cheapest labor on earth, and by processes not as yet developed, it still has an unlimited market at a definite price. There is no reasonable fear of disturbance by the finding of bountiful deposits, or by cheapening the cost of production due to devaluation or by the development of any easy processes for direct, or for byproduct, production. Tabulations show world production from

1492 to 1931; political distribution of production; yield and profit from mines in various gold fields; sales and dividends of other industries; gold consumption in the arts, and India's absorption of gold. Silver is also discussed by the author.

Milling and Concentration

Soap Flotation of the Nonsulfides. By Will H. Coghill and J. Bruce Clemmer. (*Tech. Pub. No.* 445. 8500 *words.*) The large number of nonmetallic ores requiring concentration is impressive. Statistically, if flotation could be applied to nonmetallics —or better called nonsulfides—as successfully as it has been utilized in sulfide ore dressing, the field of flotation would be more than doubled. Some justification for anticipating this forward move in the conservation of our natural resources is presented in this paper from the Mississippi Valley Experiment Station of the United States Bureau of Mines in cooperation with the Missouri School of Mines and Metallurgy. Nonsulfide minerals were concentrated by flotation with soap, or fatty acid derivatives. Attempts with other reagents failed. Historically, soap was one of the first reagents tried in flotation, but it was rejected on account of its tendency to float gangue. Now the property that caused its rejection in sulfide flotation recommends it for nonsulfide flotation. An outstanding observation in the fundamentals of flotation, which guided the successful concentration of 12 nonmetallic ores, was that the flotation was well under control when the mineral was flocculated and the gangue was dispersed. The following minerals were concentrated: Fluorspar, tricalcium phosphate, rhodochrosite, manganese oxides, bauxite, limestone, barite, siderite, chromite, tungsten minerals, and cyanite. Of these the flotation of the first four has been commercialized, and the laboratory work indicates that some of the others are equally amenable.

Principles of Flotation. An Experimental Study of the Effect of Xanthates on Contact Angles at Mineral Surfaces. By I. A. Wark and A. B. Cox. (*Tech. Pub. No.* 461. 22,000 *words.*) Five Australian companies and one Burma company employed the authors in Australia to study the underlying principles of flotation. It was decided to determine whether the effect of chemical collectors could be evaluated quantitatively by the contact-angle method. Although the significance or connection between angle of contact and flotation has been belittled by some writers, others believe that it is of importance and have done much experimentation. There is a close relation between angle of contact and wetting, adhesion, adsorption and flotation. The primary requisite for flotation is sufficient attraction between mineral and air for air to be able to replace part of the water at the mineral surface. In the research the bubble machine of Taggart, Taylor, and Ince was used. One of the major difficulties in estimating contact angles is that it varies according to the condition of the surface of a mineral particle. The minerals used were prepared by grinding and polishing. Xanthates were prepared and purified. Potassium ethyl xanthate at a concentration of 25 mg. per liter and a pH value of 6 to 6.5 was mostly used. Characteristic angles of contact were 59° for galena, almost 60° for sphalerite and pyrite, 58 to 62° for copper minerals, 58° for anglesite, and none for certain gangue minerals. Ten other xanthates were tried on these and other minerals and gave contact angles for certain of them ranging from 50 to 96°, compared with an average of 60° for ethyl xanthate. No contact was made by the minerals with air in distillated water. The xanthates are effective for metals. The concentration of xanthate has little effect on the contact angle, but proper choice of reagent concentrations is difficult. It was found that activators as copper sulfate were needed for sphalerite and other minerals. Dixanthogens affect contact angles, and the ethyl compound acts as a collector; so does carbon bisulfide. Alkalies also affect contact angles. The effect of cyanide on the contact angle was studied, also nature of gases, as sulfur dioxide and carbon dioxide, and modifying agents, as sodium silicate.

Floating Gold on the Mother Lode. By MAX KRAUT. (*Min. & Met.*, April, 175. 2100 *words.*) The author describes the experimental and commercial application of flotation to gold ores at several well-known mines, presenting details and figures on the preparation of the ore and the consumption of reagents. He reports the discovery that the quantities of reagents constitute an important factor in that recovery improved and grade of concentrate was raised as the quantities were gradually decreased. He concludes that flotation of gold ores must be justifiable on economic grounds and in this respect all indications are that the process is holding its own. The choice of disposal of concentrates depends entirely upon local conditions. Whether it is more profitable to ship to a smelter or treat locally by cyanidation or some other method can only be decided for each individual case, depending on grade of concentrate, distance to smelter and comparative cost of smelter treatment and local treatment by cyaniding or other methods.

Advantages of Washing Flotation Feed. By A. L. ENGEL. (*Min. & Met.*, May, 234. 1600 *words.*) Washing flotation feed should be given consideration when the nature of the ore makes ordinary treatment difficult. There would be better results from using the conditioning reagent and removal of colloids and harmful material would enable more efficient use of other reagents. Flotation control would be simplified, since the ore would be well mixed.

Gold Milling Developments in Northern Ontario. By WILLIAM F. BOERICKE. (*Min. & Met.*, September, 391. 12,000 *words.*) A detailed description of milling work at the larger mines at Kirkland Lake and Porcupine. Flow sheets are presented of the mills at Wright-Hargreaves, Lake Shore and Dome. Special attention is given to a detailed description of the new McIntyre mill at Porcupine, with a full account of the flotation practice adopted there to replace the all-cyanidation work formerly used at the old mill. Another important feature of the paper is the description of the flotation work at Lake Shore mines, where the filter tails, instead of being discarded and sent to waste, are subjected to flotation, the concentrates reground to minus 600 mesh and subsequently cyanided to provide an additional important gold recovery. A description of the unique blanket plant at Dome mines is also included, with details of the operation. A comparative table gives a summary of Porcupine milling practice, with head assays, extraction, and method of treatment at four mills, and a second table gives comparative milling costs at representative mills in each camp. General comments are offered on character of ore treated, labor and power costs, mill construction and reagent consumption. Several new types of equipment encountered in the mills are briefly described.

A Mill for the Small Gold Mine? By JOHN A. BAKER. (*Min. & Met.*, May, 209. 9000 *words.*) An article for those who are more or less inexperienced in the gold-mining industry. The questions considered, with explanatory statistics, are (1) whether any mill at all is justified, (2) how large the mill should be and what profit may be expected, (3) what milling methods should be used. Amalgamation, stamping, cyaniding and flotation processes are discussed.

Flotation in the Treatment of Gold Ores. By ROBERT L. KIDD. (*Min. & Met.*, September, 405. 1200 *words.*) Three general type of ores were studied: (1) quartzite containing 0.26 oz. gold per ton in which no base metals are present; (2) sulfide ore (2.04 oz. Au; 1.31 oz. Ag) containing pyrite as the only sulfide present and the gold free; and (3) ozidized ore (1.10 oz. Au; 0.70 oz. Ag) very little sulfide iron present. In type 1 the gold is free and bright, in types 2 and 3 the gold is abundantly present as free gold, less than 20 per cent of which is tarnished. The gold particles occur in sizes from approximately 100 mesh down to extreme fineness but the major portion lies between 100 mesh and the equivalent of 800 mesh. Most of the gold is liberated by grinding to minus 65 mesh. With either straight flotation or combined flotation and gravity concentration, finished concentrate assaying over 1000 oz. of gold per

ton can be produced. The tarnished gold did not respond to straight flotation but after it had been concentrated by tabling the flotation tailing, and reground, it floated readily. There was no difficulty in floating bright free gold either in the presence or absence of pyrite.

Technical Advance on the Mesabi Iron Range. By RUSSELL H. BENNETT. (*Min. & Met.*, June, 261. 3800 *words.*) The caterpillar-mounted electric shovel has meant increase in dipper capacity, reduction in time for digging and swinging cycle, and the elimination of a large pit crew. Electric haulage is being introduced in open-pit mines and the possibilities of conveyor-belt transportation are being studied. But in ore concentration lies the future life of the Mesabi Range. The methods of treatment used are washing, crushing and screening, sintering and drying. Magnetic roasting offers the most promising field for radical advance in the art of ore concentration on the Mesabi. Its application is probably impossible at the present cost of supplies and at the present price for the product, but if natural gas should be brought in the advent of the process would be greatly hastened and the cost of the product materially reduced. Development on the Mesabi Range is sure to continue unless the life of the industry is curtailed by an increasing burden of taxation.

Iron and Steel Division

On the Rates of Reactions in Solid Steel. By EDGAR C. BAIN. (*Trans.*, vol. 100. 16,000 *words.*) A basis for evaluating the contribution of any alloy addition toward the development of deep-hardening or air-hardening quality in steel has been suggested, dependent upon the effect of the addition to retard transformation in the 600° to 500° C. range. This is the only fundamental property involved in securing the final quality of hardening without drastic quench. An element, manganese, contributing very greatly to this effect has been contrasted with an element, nickel, which shows only a little retarding effect upon the transformation in the significant range. A series of similar steels differing substantially only in manganese content has been examined for rate of transformation, and a total range of transformation velocity covering about 1 to 1,000,000 is revealed. Graphite has been found to be the stable form of carbon in 0.50 per cent C, 3.5 Ni steels free from significant amounts of other elements. Nevertheless, cementite, because of its high speed of formation, is found to form in large proportion even though it is less stable than graphite. The relative velocities of the reaction producing carbide and that producing graphite are considered to be the cause of the presence of carbide alone in nickel steels in common use. A stable condition of three phases, ferrite, austenite and carbide, in equilibrium is found to exist in certain manganese steels. This condition ultimately results after long heating at a proper temperature whether the original metal be austenitic or pearlitic. The state of three-phase equilibrium is reached, however, much more quickly when the starting material is austenite. From the observation that "white martensite" is converted into a coarse ferrite-carbide aggregation thousands of times faster than is austenite of the same composition, it is concluded that martensite cannot be an intermediate state in the austenite-pearlite reaction. Almost all of the reactions so far discussed reveal a velocity pattern very nearly that of simple first order chemical reactions. There are, however, reactions in steel which begin at very high velocity and then soon slow up to a point at which it is barely possible to find any further change whatever. Two examples of such reactions are presented; the solution of carbide in austenite at a temperature above A_{cm}, and the coalescence of carbide during tempering. In both of these reactions, carbon diffusion begins with steep concentration gradients and short paths of migration, but such paths rapidly lengthen and the gradients become less and less pronounced as the process continues.

Sinter in Blast-furnace Burdens. By ROBERT MCCLURKIN. (*Trans.*, vol. 100. 3500 *words.*) The use of sinter in the iron blast furnace increases the daily tonnage,

owing in part to the enrichment of the burden and the reduction in amount of flue dust produced. Sinter gives a burden that is more easily smelted and handled than a burden composed of raw ore, owing undoubtedly to the porosity and physical size of the sinter and, incidentally, to the chemical condition of the iron in the sinter and the absence of combined water.

Sintering Economics. By Perry G. Harrison. (*Tech. Pub. No.* 480; also *Trans.* vol. 100. 4000 *words.*) Geographical location and character of material sintered so radically affect sintering costs that equally efficient large-scale continuously operated plants will show costs of from 65¢ to $1.50 per ton. By reason of low power and fuel costs sintering at steel or blast-furnace plant locations is cheaper than elsewhere. However, due to possible freight and royalty saving, the most economical location for a sintering plant treating ores having high moisture may be at the mine. Lake Superior ores having in excess of about 15 per cent free and combined moisture are most economically sintered at the mine, those having less moisture at Lake-front furnace. The principal reason for sintering an ore is to obtain improvement in structure. Crude ores that consist of a mixture of coarse material and fines are most economically treated by the "sinter-dried" process developed during the last year by the Evergreen Mines Co. at Cosrby, Minn. This process consists of first crushing an ore to eliminate all large lumps and boulders, and screening at ¼ in. The fines are sintered and the hot resultant sinter recombined with the coarse ore. The heat of the sinter drives off part of the superficial moisture from the coarse ore; i. e., the ore is "sinter dried." The structure of sinter-dried aggregate is ideal for furnace use.

Effect of Small Percentages of Chromium on the Quality of Cast Iron. By C. O. Burgess. (*Tech. Pub. No.* 492. 5000 *words.*) Small additions of chromium (0.25 to 1.00 per cent) were made to cast irons of varying composition to investigate the effect of these additions on the quality of the resultant irons. The change in hardness of cast iron with increasing chromium content was evaluated and the influence of initial hardness on this change was considered. The strength of all the cast irons was definitely increased by the chromium additions and no sacrifice in deflection was evident up to a chromium content of 0.50 per cent. The use of small chromium additions was found effectively to prevent loss in hardness and strength of cast iron on increase of section. The structural stability of the irons at high temperatures and their resistance to growth was found to be definitely increased by the presence of chromium. Additions of chromium were found to result in the production of a pearlitic type of cast iron, explaining to some extent the higher quality of the irons containing this element. The machinability of the irons containing small amounts of chromium was not adversely affected even though the hardness was moderately increased.

Some Effects of Temperature and Iron Oxide in the Manufacture of Basic Open-hearth Steel. By W. J. Reagan. (*Tech. Pub. No.* 469; also *Trans.*, vol. 100. 5000 *words.*) This paper details briefly the results obtained by an effort to control the iron oxide content of bath and slag in basic open-hearth practice. Fast melting is conducive to low FeO content. Competent furnace operators are also an important factor in FeO control. The effects of temperature and iron oxide on residual manganese are charted in detail. Low FeO and high temperatures combine to enable final manganese additions to be kept at a minimum. Basic slags with a high SiO_2 content usually show a low FeO content. Furnace temperatures depend largely on furnace life and curves given show the effect of furnace life on open-hearth rejections.

Inclusions—Their Effect, Solubility, and Control in Cast Steel. By C. E. Sims and G. A. Lillieqvist. (*Tech. Pub. No.* 453; also *Trans.*, vol. 100. 12,000 *words.*) Numerous observations and close checking on specimens of cast steel have revealed a striking and consistent relation between the ductility of the steel and the type and distribution of the inclusions present, other factors being equal. Subnormal ductility

has been found to be directly due to inclusions which form as a part of a eutectic in the steel. The substance of which the normal or natural inclusions of steel are formed has been found to be soluble in the molten steel. The best evidence of solubility is the variation of inclusion size with the rate of solidification. If the inclusions are precipitated before or at the beginning of solidification, the steel will be ductile; if they precipitate as a eutectic, the ductility will be low. Whether the inclusions precipitate early or late in the solidification period is mainly dependent on the iron oxide content.

Analyses of Inclusions in High-carbon Tool Steels. By Haakon Styri. (*Preprint.* 3500 *words.*) The electrolytic method for separation of inclusions in steel has given satisfactory results when a membrane is arranged around the anode to collect the undissolved particles. Suitably dense paper filters attached to the bottom of a glass tube may serve better for many purposes than collodion bags, because they permit direct filtration. A variety of electrolytes may be used, provided care is taken that the cathode chamber is kept slightly acid (0.1 per cent or less), so that hydroxides are not formed on the membrane, and that the anode chamber does not get too acid so that precipitated particles will be attacked to a disturbing degree. When it is desired to keep carbides away from the residue, the steel may be quenched from a temperature sufficiently high to bring the cementite in solid solution. Determination of sulfur by the volumetric method, giving practically complete recovery of the sulfur present in the steel, indicates that sulfur is present as sulfide, probably of iron and manganese. Short-time treatment of residue with weak acids does not seriously affect the sulfides or manganese but does attack some of the iron. Seventy-two hours' digesting with 25 per cent sodium hydroxide brings all the silica into solution and also attacks the sulfide. Digesting with 10 and 20 per cent sodium citrate brings a large amount of iron, but only a smaller amount of silica, sulfur or manganese into solution, indicating that the manganese is not present as manganese oxide but probably mostly as silicate. The amount of iron found in the residue from heat-treated samples is rather constant and probably a large part of it is bound to the slag particles; the rest is metallic iron or, to a small degree, hydroxide. The determination of sulfur by electrolysis and treatment of the residue by the Eschka method is so convenient that it may be used for routine gravimetric analyses of sulfur where time is of little importance. The total proportion of silica is very constant, except in one sample where the quantity is so small that great error in analyses may be expected. Only a small amount of silica is apparently present as colloidal silica, as treatment with hot 3 per cent sodium carbonate takes some in solution.

Tensile Properties of Rail Steels at Elevated Temperatures. By G. Willard Quick. (*Preprint.* 10,000 *words.*) In continuation of previously reported work on a study of transverse fissure failures in railroad rails, a further series of tensile tests at elevated temperatures has been made on rails representing different conditions of manufacture and service, to determine the extent to which secondary brittleness occurred in these materials. Slower cooling from the hot saw tended to reduce the secondary brittleness at some sacrifice of tensile strength. No difference in tensile strength or secondary brittlness was found between fissured and unfissured rails that had been subjected to the same service. No appreciable difference in ductility was found in rails made at a certain mill when their rails were yielding poor service records and rails made subsequently which had improved service records; although secondary brittleness was not marked in either series, the ductility was rather low in the higher temperature range. Shatter cracks were found in all rails examined that had failed in service. Secondary brittleness was manifest in both used and unused sorbitized rails. Shatter cracks were found in both.

Resistance to Impact of Rail Steels at Elevated Temperatures. By G. Willard Quick. (*Preprint.* 5000 *words.*) Charpy impact tests were made in the tempera-

ture range 20° to 700° C. on V-notch specimens cut transversely and longitudinally from a medium manganese rail; and on longitudinal specimens cut from two standard rails, one cooled normally and the other cooled slowly after leaving the hot saw; two heat-treated rails and one untreated comparison rail; and a commercial bar stock steel containing 0.60 per cent carbon. The energy absorbed in breaking the specimens from all these materials increased as the temperature increased from 20° to about 400° C., then decreased to a minimum of about 600° C., after which it increased rapidly to 700° C. Tensile tests on the rail steels, reported previously, had shown a marked decrease in elongation and reduction of area for all of the rail steels between 400° to 700° C., with the minimum values at 550° or 600° C., but not for the bar stock. The tensile test had also shown somewhat higher ductility in the secondary brittle range for the slowly cooled rail and the heat-treated rails than for the normally cooled and the untreated comparison rail, but the impact test did not distinguish between them. The path of the fractures in the impact specimens was transcrystalline at all temperatures. It is believed that the minimum values for impact strength occurring at 550° or 600° C. may be ascribed to the same phenomenon as the low ductility disclosed by the tensile tests of the same temperatures, namely that of secondary brittleness rather than to the phenomenon of blue brittleness which some believe occurs at a higher temperature in dynamic tests than in tensile tests.

Critical Studies of a Modified Ledebur Method for the Determination of Oxygen in Steel. By B. M. Larsen and T. E. Brower. (*Preprint;* also *Trans.*, vol. 100. 22,000 *words.*) The use of a nearly vacuum-tight hydrogen train with a sample consisting of thin steel millings suspended in a steel foil container in a transparent silica combustion tube heated to 1100° C., with two weighing tubes for water vapor and a reduced nickel-thorium oxide catalyst to convert CO and CO_2 to CH_4 and H_2O, was found to give an apparent qualitative recovery of all oxygen-bearing gases evolved from the sample, with a true blank value of only about 0.2 mg. H_2O per hour. Surface oxygen on millings can be removed by a preliminary heating at 500° to 550° C. for one hour. The "diffusible oxygen" so determined (to an apparent accuracy of about ± 5 to 10 per cent) is evolved from thin layers of steel in a manner which follows closely the simple laws for the diffusion of heat or dissolved substances in solution. There were indications that certain oxides, such as SiO_2 and MnO which in the pure state cannot be reduced by hydrogen at 1100°, will be reduced slowly under these conditions when in contact with certain metals such as platinum or iron. Comparative analyses gave some indications that "total oxygen" determinations by the vacuum-fusion method may give low results for oxygen in certain steels.

Determination of Oxygen, Nitrogen and Hydrogen in Steel. By J. G. Thompson. (*Tech. Pub. No.* 466. 9700 *words.*) The amounts of oxygen and nitrogen found in normally clean, sound steel will vary from a few thousandths to a few hundredths of one per cent. The amount of hydrogen usually is negligible. The different methods which have been proposed for the determination of oxygen, nitrogen and hydrogen in steel are discussed in some detail. In the determination of oxygen, no one analytical method yields complete information. Residue methods, including electrolytic methods, determine certain insoluble constituents, but fail to determine more soluble ones. Hydrogen reduction methods are practically limited to the determination of FeO. Macroscopic and microscopic methods are limited in their application. The vacuum-fusion method determines the total oxygen content of many steels but does not distinguish the forms or combinations present. Any of the methods may be valuable for comparisons of like materials, but complete information, if it can be obtained at all, ordinarily comes only from a combination of two or more of the methods. In the determination of nitrogen, satisfactory results usually can be obtained by either of two methods, the solution-distillation method or the vacuum-fusion method. Hydrogen is rarely found in ferrous materials in amounts in excess

of about 0.001 per cent. If the determination of hydrogen is desired, it can be included in a vacuum-fusion determination.

Equilibrium Diagram of Iron-manganese-carbon Alloys of Commercial Purity. By E. C. BAIN, E. S. DAVENPORT, and W. S. N. WARING. (*Tech. Pub. No.* 467; also *Trans.*, vol. 100. 12,000 *words.*) Described in this paper is the constitution at substantial equilibrium of 36 commercially melted alloys of iron, manganese, and carbon, covering the iron-rich range to 1.5 per cent carbon and 15 per cent manganese. Principal characteristics of the three-component equilibrium diagram are: (1) the existence of a definite three-phase temperature zone (ferrite-austenite-carbide), and (2) the general lowering of transformation temperatures with increase in manganese and constant carbon, and (3) the elevation of these same critical temperatures with increase in carbon with constant manganese. Unusually long maintenance at temperature is required to attain equilibrium. The three-dimensional equilibrium diagram is depicted by vertical constant-manganese and constant-carbon sections.

The Intermediate Phases in the Iron-tungsten System. By W. P. SYKES and KENT R. VAN HORN. (*Preprint.* 7000 *words.*) This investigation confirms the existence of an intermetallic phase approximating in composition the formula Fe_2W which has been proposed by Arnfelt. Evidence furnished by X-ray patterns, microstructures and chemical analyses indicates that this phase is formed at about 1040° C. by a peritectoid reaction betwen Fe_3W_2 and the iron-rich solid solution. Diffraction patterns of these two intermetallic phases isolated by electrolytic solution agree closely with those published by Arnfelt. A series of alloys in the range of composition between 80 and 90 per cent tungsten was treated in an attempt to form the *X* phase recently reported by Takeda. No indication of such a phase was observed in either the microstructures or the diffraction patterns.

Effect of Vanadium in High-speed Steel. By A. B. KINZEL and C. O. BURGESS. (*Tech. Pub. No.* 468; also *Trans.*, vol. 100. 3300 *words.*) A study has been made of the effect of increasing percentages of vanadium on high-speed steels of the usual carbon content, as well as of the effect of such increased percentages on standard and cobalt-bearing high-speed steels in the presence of higher carbon. Satisfactory forgeability and improved cutting properties were obtained in both the standard and cobalt high-speed steels when the vanadium and carbon contents were increased in the proper amounts. An 18 per cent tungsten, 4 per cent chromium, 5 per cent vanadium, and 1.5 per cent carbon alloy was produced, with excellent forgeability and markedly improved cutting properties.

Resistance of Iron-aluminum Alloys to Oxidation at High Temperatures. By N. A. ZIEGLER. (*Tech. Pub. No.* 450; also *Trans.*, vol. 100. 2800 *words.*) Iron-aluminum alloys with 2, 4, 6 and 8 per cent aluminum, and carbon ranging from practically zero to about 0.4 per cent, were investigated. No difficulty was experienced in forging any of them. The study of their resistance to oxidation at high temperatures showed that even 2 per cent aluminum caused a considerable improvement; with 4 per cent aluminum, this improvement becomes appreciable; with 6 per cent aluminum, these alloys are only slightly inferior to a standard nickel-chromium alloy (corresponding to nichrome No. 1, and chromel A); and with 8 per cent aluminum they are just as good. Carbon content, on the other hand, shows little, if any, effect on the oxidation resisting properties of these alloys.

Effect of Heat Treatment on the Corrosion Resistance of Stainless Iron. By CLARENCE G. MERRITT. (*Preprint; also Trans.*, vol. 100. 5000 *words.*) Attention is called to the fact that heat treatment is an important factor in the corrosion resistance of stainless iron. The alloy quenched in oil and subsequently tempered in the neighborhood of 1000° F. is quite brittle and susceptible to corrosion. This fact, as well as characteristics observed in partly hardened steel quenched from too low a temperature, probably explain failures which have remained mysteries and have

narrowed the field of application of an alloy which has quite remarkable proper when heat-treated with a full knowledge of its metallurgical behavior. This pa details the effect of various tempering temperatures on the corrosion resistar tensile characteristics, impact strength, and microstructure of the material. ⁊ conclusion is reached that the dangerous tempering range is concurrent with breaking down of chrome martensite, the formation or coarsening of a fine, submic scopic carbide precipitate which, when coalesced to microscopic size by higher temp ing, is not harmful, but produces a structure characteristic of the material wl extremely tough and resistant to corrosion.

A Quantitative Method for the Estimation of Intercrystalline Corrosion Austenitic Stainless Steels. By J. J. B. RUTHERFORD and ROBERT H. ABORN. (*I print;* also *Trans.*, vol. 100. 3200 *words.*) Investigation of the unique type of in crystalline corrosion which may occur in austenitic stainless steels following expos of the metal to a temperature within a range corresponding approximately to 10 to 1500° F. (540° to 815° C.) has led to the development of a satisfactory methoc measuring the extent of such corrosion. The method is based on the measuremen the change in electrical resistance of a specimen occurring during the intergranu corrosion attack. The important details of procedure are described, and in addit a quantitative study of the effect of several factors upon intergranular corrosion the 18-chromium 8-nickel type of alloy is presented. The results show that susc tibility to such attack increases with increasing carbon content and grain size a appears to pass through a maximum for certain combinations of temperat and duration of sensitization. The method is probably applicable to other metals a alloys subject to intercrystalline attack, such as aluminum alloys of the dura min type.

Prevention of Intergranular Corrosion in Chromium-nickel Corrosion-resist Steels. By P. PAYSON. (*Tech. Pub. No.* 464; also *Trans.*, vol. 100. 12,000 *wor*ı Since welded structures of the popular 18-8 corrosion-resistant steel are subject intergranular corrosion in the material adjacent to the weld, considerable thou has been given to the prevention of this type of attack. The present paper descri a method of accomplishing this. It was found that intergranular corrosion could prevented by adding to the 18-8 composition, sufficient quantities of silicon, mol denum, or other ferrite-forming elements, so that the steel would contain two pha austenite and ferrite, when quickly cooled from 1850° to 2100° F. The explanat of the resistance to intergranular corrosion found in the two-phase steels seems lie in the fact that the precipitated carbides in the reheated two-phase steels found mainly in the ferrite rather than in a concentrated form along austenite-gı boundaries, as in the regular 18-8. Additions of titanium to regular 18-8 produ resistance to intergranular attack even though a two-phase structure was not de oped, but no explanation has been offered. The author gives a résumé and critici of the prevailing opinions on the cause of intergranular corrosion.

Electrochemical Potentials of Nitrified Steels. By SHUNICHI SATOH. (*T Pub.* 447. 7000 *words.*) The author measured the single potential differences nitrified or non-nitrified special steels containing chromium, aluminum, titani and zirconium in distilled water, sea water, saturated copper sulfate solution, normal ferrous sulfate solution. He found that special steels, when nitrified, becı nobler about 0.6 volt in distilled water and about 0.2 volt in sea water, but becı baser about 0.07 volt (owing to atomic hydrogen) at first and finally about 0.03 vol normal ferrous sulfate solution. In saturated copper sulfate solution, nitrified spe steel has a positive potential of about 0.2 volt. In distilled water, nitrified st become black and never rust to form red hydroxide of iron. In normal fer sulfate solution, the single potential of nitrified iron is so greatly influenced that presence of a small amount of ferric ion will produce a potential difference of ab

one volt. In measuring the single potential of various nitrided steels, the author found that porous iron may be produced in the outer layer of steels as the result of nitrification and of immersion in ferrous sulfate solution.

Pressure Welding Low-carbon Steels with Theoretical Considerations on the Mechanism of Such Welding. By C. R. Austin and W. S. Jeffries. (*Tech Pub. No.* 451. 19,500 *words.*) The paper describes the results of experiments on the experimental welding of low-carbon steels under various controlled conditions. Details are given of the development of the necessary technique for the production of pressure welds at 1350° to 1450° C. in both oxidizing and in reducing atmosphere and also in vacuo. The methods used to determine the quality or amount of welding effected on butt-welded rods include the tensile test, impact test, microscopic and macroscopic examination, and electrical resistivity determinations. About 300 welds were produced and examined, and the results have been tabulated so as to define the atmosphere and temperature of welding, pressure of welding, the ultimate tensile strength, the elongation per cent, and the nature of the fracture. Most of the welds were made in air, in which cases they broke at least partly in the junction line of the weld when pulled in tension. The resulting fracture is shown to be composite and to consist of areas of true weld exhibiting a fiery fracture, and areas of "sticking" exhibiting both a true grey fracture and complete absence of welding. All these welds showed poor resistance to shock (impact) test, but they frequently gave tensile values well above those which could be accounted for from determinations of the true area of weld. A theory has been outlined to show that the presence of a thin oxide film prevents welding in certain regions, but confers high resistance to tensile rupture. It is well known that thin films are capable of carrying high tensile stresses, and on account of their thinness they transmit little or no shearing stresses. Impact or sudden transverse stresses produce failure regardless of the thinness of the film if the film material is anything but ductile. Several experiments were carried out with a view to examining the validity of the hypothesis. These included pressure welding in air with a superimposed translatory movement of the contact areas, pressure welding in hydrogen without this translatory movement, and extremely light-pressure welding in vacuo. It is considered that the evidence produced from this experimentation confirms the theory outlined by the authors on the mechanism of pressure welding.

Metallic Coatings for Steel. By Marvin J. Udy. (*Min. & Met.*, April, 173. 2100 *words.*) Metallic coatings are generally applied to steel to improve its appearance, to resist corrosion, and to resist wear and abrasion. Coating for resistance to wear has become of importance with the development of chromium plating. Coating for improvement in appearance is practically confined to electroplating. Where resistance to corrosion is the primary object, and appearance is secondary, zinc, tin and cadmium can be efficiently applied directly to steel. The resistance to corrosion obtained by any single metal coating depends somewhat on the particular metal, but to a larger extent on the perfection of the coating process, which should insure freedom from pinholes, proper thickness, and adherence of the coating to the steel. Resistance to abrasion is not satisfactorily secured from metallic coatings because any economic metallic coatings must necessarily be thin. Soft materials offer more resistance to abrasion than hard materials. Hard materials like chromium are ground with relatively soft abrasives. Chromium does not resist abrasion so much from its hardness as because of its low coefficient of friction. The two most widely used methods of applying metallic coatings to steel are electrodeposition from solution and hot application, such as galvanizing, Sherardizing, or spraying. Thickness of coatings and uniformity are secured by proper technique in operation. Cleaning is vital to perfect adherence.

Economic Notes on Steel-making Alloys. *See* Mining Geology.

Nonferrous Metallurgy

Reverberatory Smelting of Raw Concentrates at the International Smelter, Arizona. By P. D. I. Honeyman. (*Tech. Pub. No.* 456. 3500 *words.*) Changing trends in concentration, with the production of high-grade concentrates, present a problem to the modern smelter the solution of which may lie in raw charging. Such a line of attack has been followed at the Miami plant of the International Smelting Co. with four years' development of the idea, following its introduction at Cananea. Charge containing 11 to 13 per cent moisture is taken to the furnace in special hopper-bottom cars, dumped into bins, and fed to the furnace by a pan feeder and drag-chain conveyor system. Gross fuel consumption is higher than when smelting calcine, but capacity of the furnace is limited only by the possibility of obtaining complete combustion of the fuel burned. Higher waste-heat recovery, however, to some extent offsets high fuel consumption. In the operation a normal sulfur elimination is obtained and reactions denoting reduction of magnetite contained in molten converter slag are plainly noticeable. Absence of magnetite accumulation within the furnace is a feature of raw charging. Side-wall skimming and front-wall tapping have proved favorable. The particular advantage of raw charging is a decrease in the amount of dusting, which in turn results in lower losses, simplification of operation, and long furnace life with a minimum of repairs.

A Comparison of the Use of Various Fuels in Copper-refining Furnaces. By E. S. Bardwell. (*Tech. Pub. No.* 457. 9000 *words.*) The reverberatory copper refining furnaces at the Great Falls Reduction Works of the Anaconda Copper Mining Co. furnish a unique opportunity for comparing the results obtained with various fuels, as these furnaces have been successively operated with lump coal on grates, pulverized coal, oil and natural gas. The comparison of pulverized coal, oil, and gas is of particular interest because these fuels have been used on the same furnace under identical conditions. It is shown that the tonnage remaining constant, pulverized coal, oil, and natural gas cannot be substituted one for the other on an equivalent heat-unit basis. An analysis of combustion conditions shows that with the pulverized coal 46.7 of the heat value is available to supply radiation losses and the heat necessary to do the work in the furnace chamber. With the oil fuel used, 44.7 per cent of the heat value of the fuel is thus available, but with gas only 41.9 per cent. On this basis it might be thought that pulverized coal would be the most efficient. Using pulverized coal, however, the B.t.u. requirements per pound of copper produced were 1627, with oil 1441, and with natural gas 1670. The discrepancy between the heat-unit requirement for pulverized coal and oil is explainable in part at least on the basis of the insulating effect of the ash which settles on the molten bath. The heat-unit requirement for natural gas was higher than expected though the theoretical figure seems possible of attainment.

The Messina Stationary Basic Copper Converter. By R. G. Knickerbocker. (*Tech. Pub. No.* 458. 3200 *words.*) The copper smelter and refinery of The Messina (Transvaal) Development Co., Ltd., at Messina, South Africa, began operations late in 1922. The plant was built for the production of copper by the Nicholls-James process, but after a short trial this was abandoned and the reaction furnace was remodeled to serve as a stationary converter. This paper describes the design, operation and economy of the smelting and refining practice that resulted, and concludes that the revised Messina practice demonstrates that high-grade copper matte can be economically converted to blister copper in a stationary converter with the intermittent use of additional heat and without the machinery used in the standard converting practice; that refined copper meeting A. S. T. M. specifications can be produced from molten Messina blister copper without allowing it to solidify before charging to the refining furnace, and that the Messina practice is good for small or

medium sized copper producers where the available labor is cheap and not highly skilful.

Development of the Leaching Operations of the Union Minière du Haut Katanga. By A. E. WHEELER and H. Y. EAGLE. (*Tech. Pub. No.* 459. 14,000 *words.*) The steps in the development of the Union Minière du Haut Katanga leaching operations are given in chronological order. These cover the period from 1914 to 1930, and include the early mine inspection and experimental work, the gravity concentrator, the large-scale experimental leaching plant, and the new 30,000-metric-ton Panda leaching plant. Although the Katanga ores are amenable to sulfuric acid leaching, their peculiar properties necessitated a departure from established copper leaching practice. The process is continuous instead of batch, leaching is effected by agitation instead of by percolation, sand tailing is washed separately in Dorr classifiers arranged for counter-current washing, slime tailing is washed separately by counter-current decantation in Dorr thickeners, and copper is recovered from solution by electrolysis in large tanks of low circulation rates. Interesting developments of the work are the long electrolytic tanks, continuous agitation of comparatively coarse material, and the scheme of purification for limiting the iron content of the solutions to the tank house. Operating data are included.

Development of Gun Feed Reverberatory Furnaces at the Garfield Plant of the American Smelting & Refining Co. By R. A. WAGSTAFF. (*Tech. Pub. No.* 471. 2500 *words.*) A new type of feeder, developed at the Garfield plant of the American Smelting & Refining Co., introduces dusty, fine calcine into reverberatory furnaces. The apparatus consists of a disappearing gun feeder, with a water-cooled nozzle, that can be projected into the furnace through openings in the side walls. The charge enters the furnace at an angle, under the main gas stream of the combustion zone, at a velocity sufficient to distribute it in a thin, even blanket over the highly heated bath.

The feeding of the charge in this manner has four smelting advantages: lower metal losses; rapid smelting conditions; economical heat transfusion; longer furnace life, due to less fluxing action on the brickwork from dusting. The number of guns necessary depends on the size of the furnace and the amount of material to be smelted. The guns are staggered on opposite sides, to allow even distribution of the charge. At present three furnaces at Garfield are equipped with these feeders, and the results have been gratifying.

The New Lead Refinery and Its Operation at the Bunker Hill Smelter. By A. F. BEASLEY, J. B. SCHUETTENHELM, and J. W. JOHNSON. (*Preprint.* 5000 *words.*) A detailed description of the new lead refinery at the Bunker Hill smelter and its operation. Increased incoming ore tonnage, early in 1930, required a choice between additional furnaces and kettles or rebuilding the entire plant with equipment of increased capacity. The latter was adopted, mainly because of the more economical operation of larger units. Comparative labor and fuel costs of the old and new refineries show considerable operative cost reduction from the installation of larger equipment. Local conditions necessitate certain deviations from standard practice. Chief among these is the removal of gold and silver from the bullion by differential separation. This involved a consideration of present bluestone plant and silver refinery capacities. These, should standard practice be followed, would require a 200 per cent expansion to handle adequately the large tonnages of silver-lead concentrates received, two or three months per year, from the Yukon territory. The additional equipment required would represent an investment of about $85,000 and would remain idle from eight to nine months of the year.

Sintering Zinc Ore at Rosita, Mexico. By H. R. MACMICHAEL. (*Tech. Pub. No.* 455. 2000 *words.*) Paper describes sintering plant in operation two years, designed for 9000 tons per month of fine flotation concentrate preroasted to about 7 per cent sulfur. Such ore sinters only about one-half as fast as coarser concentrate thoroughly

preroasted, so required sintering capacity is unusually great. Construction and operating costs were reduced by specially designed machine three times usual size, with 300-hp. fan. Outline drawings show arrangement of plant and photograph shows main sintering floor. Baghouse is used for collection of fume. Development of the large sintering machine is described. Using cast steel pallets equipped with Timken roller bearings and automatic wind-box seal, sinter machine proper has required practically no repairs after two years' operation, except upkeep on grate bars.

Rare Minerals and Metals

The Magneto-optic Method of Analysis with Particular Reference to the Detection of Elements 85 (Alabamine) and 87 (Virginium) and the Heavy Isotope of Hydrogen. By FRED ALLISON. (*Preprint.* 7000 *words.*) The magneto-optic method of analysis and the apparatus are described in detail. The experimental results may be summarized: every inorganic compound of the more than 200 compounds studied, including more than 50 metallic elements, produces its characteristic minimum, or minima, of light intensity which are read on a scale, the scale readings being convertible into differential time lags in the Faraday effect; the minima persist until the concentration is reduced to several parts in 10^{12} of water; for compounds of a given anion the time lag is some inverse function of the combining equivalent of the cation, while for isotopes of a given cation the time lag is in some direct variation with the combing equivalent of the isotopes; the number of isotopes of the cation and their order of abundance are determined; the method has been applied in a limited manner to organic compounds, for which it has the same order of sensitivity; the presence of foreign substances has no disturbing effects. The advantages and limitations of the method as a tool in research are pointed out. In a review of the work and publications by the author and his colleagues, priority is established for the discovery of the heavy isotope of hydrogen, element 87 (virginium) and element 85 (alabamine). From studies of the physical nature of the phenomena involved, it is concluded that the results are to be explained on the basis of a time lag, probably a time lag in the Faraday effect.

Institute of Metals Division

The Age-hardening of Metals. By PAUL D. MERICA. (*Trans.*, vol. 99. 15,000 *words.*) This lecture reviews the status of age-hardening or "precipitation-hardening" in alloys, discussing in particular the theories which have been advanced to account for this important group of phenomena. The anomalous behavior of duralumin and some other systems may perhaps be fairly simply explained by assuming the possibility of hardening by segregation or "knots" of hardening atoms even prior to their arrangement on a crystalline lattice of the fully precipitated compound. The industrial importance of age-hardening alloys is emphasized in view of their several valuable properties and it is shown that their rapid development today is introducing a measure of flexibility and method in the "building" of alloys not hitherto realized. There is a two-page bibliography.

Copper-beryllium Bronzes. By J. KENT SMITH. (*Tech. Pub. No.* 465; also *Trans.*, vol. 99. 4800 *words.*) For many years it was accepted that the only way by which a "polyhedral" metal could be hardened and strengthened was by the application to it of sufficient "cold work." The discovery—with industrial application only a relatively short time ago—of the "precipitation-hardening" of one limited class of metals gave rise to intensified investigation as to whether the principle could be applied to other metals and by what means. Discovery was made that the addition of small amounts of beryllium to plain copper gave to it "age-hardening" properties of a high order indeed. This opened up a new line of investigation and thought.

Concurrent notation was made of the fact that copper containing beryllium in "solid solution" was amenable to an apparently excessive degree to the strengthening effect of "work hardening." The object of this paper is to supply quantitative information as to the above-mentioned facts, whose joint exercise led to the production—from an ingot metal containing only 1.9 per cent beryllium and 98.1 per cent copper—of sheet having a tensile strength of over 220,000 lb. per square inch.

Equilibrium Relations in Aluminum-copper-magnesium and Aluminum-copper-magnesium Silicide Alloys of High Purity. By E. H. Dix, Jr., G. F. Sager and B. P. Sager. (*Tech. Pub. No.* 472; also *Trans.*, vol. 99. 5500 *words.*) To obtain more information regarding the functions of magnesium and magnesium silicide in alloys of the duralumin type, the individual effects of small additions of these materials on the equilibrium relations in high purity aluminum-copper alloys were studied by the conventional microscopic method. The addition of 0.5 per cent magnesium slightly reduced the solubility of copper, but did not alter the form of the solubility curve appreciably. The solidus between 3 and 5.3 per cent copper was lowered to 20° to 30° C. The addition of 1.3 per cent magnesium silicide did not appreciably reduce the *apparent* solubility of copper. A decrease in the *actual* solubility may have been compensated by the formation of an aluminum-copper-magnesium-silicon constituent observed in certain phase fields. The above addition of magnesium silicide lowered the solidus extending from 0 to 5 per cent copper about 60° C.

Equilibrium Relations in Aluminum-zinc Alloys of High Purity. By William L. Fink and Kent R. Van Horn. (*Tech. Pub. No.* 474; also *Trans.*, vol. 99. 4500 *words.*) The solid solubility of C.P. zinc in electrolytically refined aluminum has been investigated by microscopic examination, hardness tests, and electrical conductivity measurements of quenched specimens, as well as by X-ray diffraction analysis at temperature. The results of the first three methods are of doubtful accuracy because the zinc phase apparently is very finely divided and room temperature aging is rapid immediately after quenching. Detection of the zinc phase at temperature by X-ray methods eliminated these sources of inaccuracy. Moreover, the cold work incident to the preparation of powdered samples greatly accelerated the attainment of equilibrium. Synthetic standards prepared by mixing definite proportions of 300-mesh C.P. zinc and electrolytically refined aluminum were used to determine the correction to be made for the excess zinc phase which is required to produce a detectable X-ray diffraction maximum. The solid solubility determined by this method was found to decrease from 13.4 per cent zinc at 250° C. to 2.7 per cent zinc at 25° C.

Equilibrium Relations in Aluminum-cobalt Alloys of High Purity. By William L. Fink and H. R. Freche. (*Tech. Pub. No.* 473; also *Trans.*, vol. 99. 4000 *words.*) High-purity aluminum-cobalt alloys containing from 0 to 8 per cent cobalt were used in this investigation. The aluminum-cobalt constituent existing in this field was separated both chemically and electrolytically and found to correspond to the formula Co_2Al_9. The hypereutectic liquidus was determined by analyzing the supernatant melt in equilibrium with the precipitated crystals of Co_2Al_9. Thermal analysis showed that the eutectic temperature is 657° C. The eutectic composition determined by extrapolation of the hypereutectic liquidus to the eutectic horizontal is 1 per cent cobalt and 99 per cent aluminum. This eutectic composition is consistent with the results of microscopic examination of specimens solidified and slowly cooled in the furnace. The solid solubility of cobalt in aluminum, determined by microscopic examination of heat-treated sheet, was found to be less than 0.02 per cent cobalt at 655° C.

The Copper-rich Alloys of the Copper-nickel-tin System. By John T. Eash and Clair Upthegrove. (*Preprint.* 14,000 *words.*) The equilibrium relationships existing in the copper-rich alloys of the copper-nickel-tin system have been investi-

gated. The alpha-phase boundary was redetermined for alloys containing from 0 to 20 per cent nickel, together with the liquidus and solidus temperatures above that field. The equilibrium conditions existing in alloys exceeding the alpha phase containing up to 31 per cent tin and 5 per cent nickel were investigated. The addition of nickel to copper-tin alloys decreases the solubility of tin in the alpha phase. The tin solubility diminishes also as the temperature is lowered. The alpha + delta eutectoid which occurs in copper-tin alloys is replaced by either the theta phase or the delta prime phase when nickel is added in amounts above one per cent. The theta phase is always homogeneous. The second new phase may be homogeneous or it may be in the form of an alpha + delta prime eutectoid depending upon the rate of cooling of the alloy. The gamma inversion to alpha + delta or to alpha + delta prime when nickel is present is raised to higher tin contents as the amount of nickel is increased.

Variations in Microstructure Inherent in the Processes of Manufacturing Extruded and Forged Brass. By O. B. MALIN. (*Preprint;* also *Trans.*, vol. 99. 3900 *words.*) A metallographic study of leaded brasses containing 57 to 60 per cent copper commonly used for brass forgings. Sections were taken from the front and rear ends of extruded rods, and photographed to show variation in grain size in different portions of the rod. Forgings were made from slugs cut from both front and rear ends of the rods; these were examined and photographed to show the effect of the original grain size of the slug upon the forging made from it. Two types of forgings were examined, one of an elongated form, and the other a short compact type.

Discussion on Some Important Factors Controlling the Crystal Macrostructure of Copper Wire Bars. (*Tech. Pub.* 485. 6000 *words.*) D. K. Crampton confirms some of Mr. De Wald's findings but feels that in giving no data on hot rolling and in not definitely correlating the initial macrostructure with the drawing performance on the final wire structure the author has overlooked an important advantage for the coarse-grained material and has assumed an advantage for the equiaxed bars that may not exist. He agrees with Mr. De Wald's facts, but not entirely with his interpretation of their significance. C. S. Harloff discusses several of the variables mentioned in the paper and sums up by saying that "no single factor or group ut factors can be said to control the adaptability of cast wire bars for any specific purpose. All factors must be carefully balanced with each other to produce bars of desired quality." C. H. Schneider believes that the relation of the weight of the mold to the weight of metal cast in it is a fourth factor to be added to those mentioned by Mr. De Wald for controlling crystal size. Morover, he does not agree that the quality of the bar for drawing fine wire depends on crystal size, which differs in different portions of the bar, but says that with rare exceptions the quality of a copper bar for this purpose is inherent in its entire length. He gives the results of five check tests, and concludes that it has not been proved that the fine macrostructure is essential in wire bars suitable for fine wire drawing. W. H. Peirce quotes from a laboratory report covering an experiment that would seem to indicate that the rate of cooling of the metal in the mold is of paramount influence. He believes that it has not been proved that fine crystal structure is preferable for wire bars for fine wire. The author's reply interprets the bases of his conclusions correlating the viewpoints brought out in the discussion.

Properties of Copper Deoxidized with Calcium. By LYALL ZICKRICK. (*Preprint.* 5000 *words.*) A high-calcium copper alloy has been found to deoxidize copper satisfactorily. Residual calcium remaining in the metal, like residual silicon, raises the annealing temperature required to produce dead soft copper; 0.05 per cent residual calcium raises the annealing temperature of copper from 250° to approximately 350° C., and when present in amounts up to 0.20 per cent, the required annealing temperature is in the neighborhood of 400° C. Owing to the very slight solubility of calcium in copper in the solid state, residual calcium does not decrease the electrical conductivity of copper as silicon does. For a specific illustration, it was found that 0.03 per

cent residual calcium dropped the conductivity of dead soft copper from 101 to 97 per cent, whereas 0.03 per cent residual silicon causes a drop from 101 to 75 per cent.

Directional Properties in Cold-rolled and Annealed Commercial Bronze. By ARTHUR PHILLIPS and C. H. SAMANS. (*Tech. Pub. No.* 491. 5000 *words.*) An effort has been made to correlate cupping and tensile tests on 90-10 brass specimens which have received rolling and annealing treatments sufficiently diverse to cover the latitude met in common mill practice. Tensile strength and elongation measurements were made on the cold-rolled material at angles of 0°, 22.5°, 45°, 67.5°, and 90° to the direction of rolling. Similar measurements and, in addition, cupping tests were made on these materials after annealing at 400°, 500°, 600°, 700° and 800° C. Rockwell hardness (16-60-B) and grain size measurements are also given, wherever feasible, to complete the data. A comparison is made between 90-10 brass and copper in regard to directional properties as manifested by tensile strength and elongation measurements.

Ears on Cupronickel Cups. By W. H. BASSETT and J. C. BRADLEY. (*Preprint.* 3000 *words.*) The edges of drawn cupronickel cups often have four ears. These may be produced either at 45° to the direction of rolling or at 90° (and 0°) to the direction of rolling. By systematically reducing the gage of cupronickel at which the intermediate anneal is introduced, a series of cups has been made in which the ears varied in regular fashion as follows: short 90° ears, long 90° ears, shorter 90° ears, no ears, short 45° ears, long 45° ears, short 45° ears, no ears. Variation in composition of the cupronickel affects earing tendencies. The length of the 90° ears increases with the final annealing temperature. Earing evidently is due to directionalism in the arrangement of the crystalline structure and to methods of rolling and annealing which bring about directional arrangement.

Some Effects of Internal Stress on Properties of Drawn Brass Tubes. By D. K. CRAMPTON. (*Preprint.* 9000 *words.*) Young's law does not hold strictly for drawn tubes, but the modulus of elasticity is maximum at zero loads and falls off continuously with increase in stress. Relief-annealed tubes more nearly approach strict proportionality between stress and strain than drawn tubes. An approximate method for comparing stress intensity and distribution in drawn tubes is described. In general, in hollow sunk tubes stresses persist well into the tube wall, whereas in drawn tubes they fall off much more rapidly. Also a harmful type of stress distribution is accompanied by high surface stress and vice versa. Polycrystalline tubes showed stresses materially higher than identically treated single crystal tubes. The simultaneous increase of hardness of surface layers originally under tension and decrease of those originally under compression is found when stress is released by splitting. Some preliminary work is reported on effect of type and degree of reduction on preferred orientation.

Structure of Cold-drawn Tubing. By JOHN T. NORTON and R. E. HILLER. (*Tech. Pub. No.* 448; also *Trans.*, vol. 99. 4000 *words.*) The structure of cold-drawn tubing has been shown to be intermediate between that of wire and sheet. If the reductions in wall thickness and in circumference are equal, the 110 axis lies along the tube axis, but otherwise the arrangement is random, which is the structure found in wire. If the reduction is essentially in wall thickness, the crystals each have a 110 axis parallel to the axis of the tube, and another 110 axis parallel to a tangent to the tube wall; in other words, the sheet type of structure. It is also shown that the structure is independent of the method of reduction, but dependent only on the dimensional changes.

Studies Upon the Widmanstätten Structure, III. The Aluminum-rich Alloys of Aluminum with Copper, and of Aluminum with Magnesium and Silicon. By ROBERT F. MEHL, CHARLES S. BARRETT and FREDERICK N. RHINES. (*Preprint;* also *Trans.*, vol. 99. 13,000 *words.*) Precipitates from the aluminum-rich alloys of aluminum

with copper, and of aluminum with magnesium and silicon, tend to take the form of plates. In the aluminum-copper alloys the precipitate is $CuAl_2$, the plates of which form parallel to the cube (100) planes in the aluminum solid solution matrix up to approximately 1.5 per cent copper. At higher copper concentrations these plates are also present, apparently in constant amount, but are accompanied by a more complex precipitation which could not be crystallographically analyzed. It is shown that the $CuAl_2$ lattice possesses atom planes, making possible the operation of the type of atomic precipitation mechanism previously postulated in these studies. The precipitate from alloys of aluminum with magnesium and silicon, ordinarily designated as Mg_2Si, could not be identified as such. This precipitate forms plates parallel to the (100) planes in the aluminum solid solution matrix in low concentrations of magnesium and silicon, but at higher also parallel to the (110) planes. Etching tests suggest that the precipitate is possibly Al_3Mg_2. The significance of these results to a general theory of the formation of Widmanstätten figures is pointed out, and possible relationships to age-hardening theories are discussed.

An X-ray Study of the Nature of Solid Solutions. By ROBERT T. PHELPS and WHEELER P. DAVEY. (*Tech. Pub. No.* 443; also *Trans.*, vol. 99. 5500 *words.*) The purpose of this investigation is to study the mechanism of solid solution, using pure silver as the solvent, and pure aluminum as the solute. X-ray measurements show that these elements have the same "shape" of atomic domains (they both crystallize as face-centered cubes) and that their atomic diameters differ by less than one per cent. The results of the present investigation point toward a chemical rather than a physical picture of the nature of solid solution; and make the older pictures merely special cases of a more general picture. The new theory has the advantage, too, that it offers a rational explanation of certain phenomena which hitherto have been only interesting detached facts. The paper shows that the solid solution of pure aluminum in pure silver lowers the lattic parameter of the silver by an amount which is proportional to the aluminum content of the solution. Saturation of aluminum in silver is reached at 5.4 per cent aluminum by weight. Further addition of aluminum gives aggregates of Ag_3Al of sufficient size, to show X-ray diffraction patterns. The experimental values for the densities of the solid solutions of aluminum in silver are somewhat lower than those calculated on the basis of a direct substitution of aluminum atoms for silver in the silver lattice. This discrepancy is greater than the combined error in the two values. Systematic examination of the various possible types of explanation for this discrepancy leaves us with only one tenable theory—that the aluminum in the solid solution is chemically combined with the adjacent silver. If this explanation is used as the basis for a general theory of the nature of solutions, the picture is found to be consistent with the known facts. So far, it has not been found inconsistent with any known fact.

On the Theory of Formation of Segregate Structures in Alloys. By C. H. MATHEWSON and D. W. SMITH. (*Preprint;* also *Trans.*, vol. 99. 2500 *words.*) Principally a discussion of Hanemann and Schröder's so-called "perfusion" figures which, on the assumption of a particular kind of diffusional process, give the geometrical location of planes along which a new phase should segregate. It is shown that the considerations advanced, particularly in the case of the alpha segregate from beta brass, require modification, and that in general the one structural feature most often met in segregate structures is the inclusion of a close-packed lattice line in the principal surface of segregation.

A Study of Segregate Structures in Copper-tin and Silver-zinc Alloys. By DANA W. SMITH. (*Preprint.* 6500 *words.*) Structures resulting from segregation of the alpha phases from the respective beta phases of the systems copper-tin and silver-zinc were investigated to determine if they were analogous to those obtained by R. F. Mehl and O. T. Marzke in the systems copper-zinc and copper-aluminum. A copper-

tin alloy containing 74.92 per cent (by weight) of copper, balance tin, and a silver-zinc alloy containing 67.76 per cent (by weight) of silver, balance zinc, were chosen for these investigations. Both of these alloys are entirely in the beta phase fields at elevated temperatures and upon cooling segregate the alpha phases. It was shown, in both systems, that the alpha phases segregated from the beta phases in the form of needles parallel to [111] directions in the beta phases, also that in the copper-tin system the alpha phase orientated itself so that a {111} plane was parallel to a {110} plane in the beta phase parent solid solution. It is postulated that the tendency of the alpha phase to form needles instead of the more common platelets is to be ascribed to the fact that a similar arrangement of atoms exists along [111] directions in the beta matrix and [110] directions in the alpha segregate, while there is poor matching of atoms between the conjugate {110} planes of the beta matrix and {111} planes of the alpha segregate. In the copper-tin system it was found that a pseudomorphic segregation of the alpha phase took place in the form of plates in positions originally occupied by twins in the beta matrix and that the twinning plane in the beta phase was probably a {133} plane.

Machinability of Free-cutting Brass Rod. By Alan Morris. (*Tech. Pub. No.* 454; also *Trans.*, vol. 99. 4000 *words.*) A machinability-testing machine of the pendulum-operated milling-machine type has been used to study the effects of various factors on the machinability of free-cutting brass rod. Results of tests reported indicate that: (1) Machinability of Muntz metal improves rapidly with small additions of lead, but less rapidly as the lead content approaches 3.0 per cent; (2) within the range of 60.0 to 63.5 per cent, the copper content exerts little, if any, influence on machinability; (3) machinability of annealed leaded brass rod is harmfully affected by the presence of beta; (4) machinability is improved by cold drawing after annealing; (5) tensile properties cannot be expected to indicate the machining quality of a rod.

The Effect of Small Percentages of Certain Metals Upon the Compressibility of Lead at an Elevated Temperatuie. By Lyall Zickrick. (*Preprint;* also *Trans.*, vol. 99. 6500 *words.*) In the manufacture of cable sheath a change from one brand of pig lead to another nearly always necessitates some readjustment of press operating conditions. Some brands of lead require higher extrusion pressures than others; for example, a copper-bearing lead requires greater pressure than a bismuth lead. This series of experiments was to obtain information about the effect of small percentages of various metals in pig lead on the compressibility at 200° C., the approximate lead-sheath extrusion temperature. Metal mixtures which cover the common lead cable-sheath alloys, such as lead alloyed with tin, antimony, calcium, cadmium-tin, were also tested. Experiments show that leads of highest purity are the softest when at a temperature of 200° C. Copper alloyed with a high-purity lead in amounts up to 0.08 per cent, or approximate eutectic composition, causes a rapid increase in the deformation pressure required, but additional amounts do not appreciably increase this pressure. Bismuth in increasing amounts produces a very gradual increase in the required deformation pressure. Data are given which show probable pressure variations when extruding different lead alloys.

Effect of Temperature upon the Charpy Impact Strength of Die-casting Alloys. By Bert E. Sandell. (*Preprint;* also *Trans.*, vol. 99. 2000 *words.*) This paper is the result of an investigation made to indicate the effect of temperature upon the physical properties of certain die-casting alloys. The Charpy impact test was chosen because of its simplicity and because of the fact that brittleness is more likely to be objectionable in die castings than hardness or other physical properties. Two types of alloys were chosen; namely, zinc and aluminum-base. Four alloys were used in all and their respective analyses were given. The testing procedure, as to temperature media, technique in handling specimens, etc., is fully explained. Average results are presented in tabular form and plotted. Two aluminum-base alloys did not show

a variation in impact strength when exposed to temperatures ranging from 0° to 500° F. The zinc-base alloys tested did show a variation in impact strength, being more brittle at the lower temperatures and reaching a maximum toughness at a temperature just below the critical temperature of decomposition of beta phase.

The Role of the Platinum Metals in Dental Alloys. By E. M. WISE, WALTER S. CROWELL and J. T. EASH. (*Preprint;* also *Trans.*, vol. 99. 21,000 *words.*) The authors have made a survey of the literature on such of the precious-metal alloys as are important to the study of the behavior of commercial dental alloys, and have summarized the results of tests on certain high-strength commercial dental alloys. Two series of quaternary alloys containing gold, silver, copper, and either platinum or palladium, were studied in detail and the effect of replacing gold with platinum or palladium upon the strength, color, and response to age-hardening heat treatments, is indicated. The authors conclude that: (1) The general nature of the age-hardening transformation is much the same whether the hardening agent is Au-Cu, Pd-Cu or Pt-Cu. (2) The introduction of either platinum or palladium, particularly the former, results in a considerable increase in solid-solution hardness, and at the same time permits the multiplication of the strength through age-hardening by substantially the same ratios as those obtainable with the weaker gold-silver copper alloys. The increase in strength resulting from the introduction of platinum or palladium is accompanied by a marked increase in melting point, whereas the increase in strength resulting from the addition of further quantities of base-metal hardeners such as nickel, copper and zinc occasions not only a decrease in melting point but also a rather serious loss in nobility. (3) The aging temperature required to produce the maximum strength in a definite time interval has been determined for each of the alloys. For 15-min. treatments this temperature is 300° C. for the gold-base alloy free from platinum or palladium, whereas it rises to about 450° C. in alloys containing large percentages of either platinum or palladium. (4) It has been demonstrated that by the introduction of the proper quantities of platinum or palladium, alloys can be produced which will develop excellent properties in spite of a considerable deviation from the optimum age-hardening temperature. In other words, such alloys possess a broad hardening range and are reasonably foolproof. (5) The results obtained by quenching from 700° C. and oven-cooling are compared with those obtained by aging at fixed temperatures, and the superiority of the fixed-temperature-aging treatment is demonstrated. This superiority is particularly evident with the palladium-content alloys. (6) The influence of quenching temperature upon the properties of the platinum-content alloys was investigated, and the great increase in strength that can be secured by increasing the quenching temperature with alloys of high platinum content is demonstrated. (7) The rates of hardening in alloys containing platinum and those containing palladium are compared, and the somewhat higher rate characteristic of platinum is shown. (8) The domain of the hardenable high-strength white alloys is indicated. (9) The economies resulting from the replacement of considerable quantities of gold by either platinum or palladium are indicated, and it is shown that in a structure of constant strength, savings in metal cost as high as 40 per cent can be secured by this means.

Magnesium: Reviewing Its Technology of Production and Use. By JOHN A. GANN. (*Min. & Met.*, April, 179. 4800 *words.*) An extensive abstract of an address to the Institute of Metals Division during the Annual Meeting of the Institute. Dr. Gann outlines the history of the magnesium industry from 1852, when Bunsen prepared this metal by the electrolysis of fused magnesium chloride in a porcelain crucible, to the date of his address. The Dow Chemical Co. entered the magnesium industry in 1916 and Dr. Gann details its experience in getting established on a commercial basis. An outline flow sheet shows the process developed. One of the outstanding effects of the growth of the magnesium industry in this country was the rapid decrease

in the cost of the metal, from $5 and more per pound in 1915, to 30¢ per pound in 1932. Tables list the more important magnesium compounds, their sources and a few of their uses; the use commercially of magnesium in five distinct fields, and the properties of magnesium alloys. The outstanding characteristic of magnesium is its extreme lightness, a quality that distinguishes it from all other engineering metals. The alloys are fabricated to practically all standard sizes of rods, angles and shapes.

Copper Embrittlement, II. By L. L. WYMAN. (*Preprint.* 5000 *words.*) The previous work on the embrittlement of copper presented by this author is extended to additional materials. These include three groups of deoxidized coppers as follows: (1) Double-deoxidized copper using silicon and calcium boride; (2) calcium-deoxidized coppers having various calcium contents; (3) double-deoxidized coppers using silicon or calcium boride with a constant amount of calcium. The results fall into a narrow range, not over 0.011-in. penetration being observed. In addition, the examinations are supplemented by wire bead tests. The results obtained are as follows: (1) The addition of calcium in small amounts (lot 2, 0.0375 per cent) gives superior qualities as regards resistance to embrittlement; (2) excessive calcium additions prove detrimental to the physical properties (bend test) of copper; (3) none of the calcium coppers appear to be detrimentally affected by the usual copper embrittlement; (4) double-deoxidized copper has shown superiority over single deoxidation, for similar or larger amounts of the same deoxidants. Lot 2 may be an exception.

Surface Effects on Assay Beads Caused by Metals of the Platinum Group. By J. L. BYERS. (*Preprint.* 7500 *words.*) During the past two decades the platinum metals have become of increasing importance in commercial alloys. As these metals generally occur in association with gold or silver, a simple method of recognizing their presence in gold and silver deposits has been needed. This paper deals with a method for their rapid determination. It depends on the effects of these alloying elements on gold and silver cupellation beads, and differentiates between the effects of the different metals. The effects of the platinum metals on the cupellation bead were found to be both structural and superficial. The structural changes were due to alteration in cell formation and variations in surface tension. The superficial effects were chiefly in the color and character of surface. By means of a combination of these variations in color and by means of other surface phenomena, it was possible to differentiate between the effects of the different platinum group metals on the cupellation bead and to estimate fairly accurately the percentage of the alloying element present.

Solubility of Gases in Metals. By V. H. GOTTSCHALK and R. S. DEAN. (*Preprint.* 7500 *words.*) In the theoretical study of metallurgical reactions, it is necessary to make certain assumptions concerning the nature of metal-gas systems. The assumption usually made is that the reaction in such systems takes place predominantly with gas that is in "solution" and that the amount of solution follows the rules which have been developed for aqueous solutions; namely, Henry's law. The purpose of this paper is to inquire whether these assumptions are the best ones that present knowledge of gas-metal systems permits us to make. In the first part the extent to which a gas will dissolve in a metal and still follow Henry's law is considered. The conclusion is believed justified that the number of molecules of gas that can dissolve in a metal and have the system follow Henry's law must be small indeed. It is therefore improbable that the first assumption concerning the predominance of these dissolved molecules in reaction is correct. It is suggested that some improvement in the correlation of the probabilities with the assumptions made in the mass law equations would be made by the use of the $\sqrt{p}$ law of Sievert in place of Henry's law. However, the amount of gas which may be held in a metal so that it follows either Henry's or Sievert's law is considerably outweighed in most metallurgical reactions by that held in gas-metal systems of entirely different structures and concerning

which little experimental evidence is available. Some speculation concerning the nature of these is offered. The conclusion is that attention should be given to the determining of the nature of gas-metal systems and that the really fruitful application of the mass law and its connected calculations will have to await a more complete knowledge of these systems than we now possess.

A Review of Work on Gases in Copper. By O. W. Ellis. (*Tech. Pub. No.* 478. 9700 *words.*) The methods used by Sievert and his collaborators and by Iwasé in their investigations of the solubilities of various gases in copper are briefly described. The results of their experiments are discussed and their discordance emphasized. Reference is also made to the recent work of Allen on the equilibrium between hydrogen and cuprous oxide. Percy's description of the characteristics of "overpoled" copper is quoted, this serving as an introduction to a description and discussion of the many experiments which have been conducted on the behavior of copper subsequent to fusion in atmospheres of different gases. The concluding section of the paper treats of the work carried out on the extraction of gases from solid and liquid metals and closes with a critique of the writer's 1928 experiments. Emphasis is laid on the need for further investigation of the phenomena of gases in metals and on the desirability of retaining an open mind as to the effects of the carbon gases and, in particular, carbon monoxide on the porosity of copper.

The Degassing of Metals. By F. J. Norton and A. L. Marshall. (*Preprint.* 14,000 *words.*) To determine how rigorous a treatment was necessary completely to remove sorbed gases from molybdenum electrodes in vacuum tubes, the degassing process in high vacuum was studied in the temperature range 800° to 2300° C., and the gases analyzed. Some work was also done on the degassing of tungsten, nickel and carbon. It was shown that the amount of gas is proportional to the mass of the sample and not to the surface area, indicating that the gas is distributed through the body of the metal. A study was made of the solubility of nitrogen in tungsten and molybdenum from 1200° to 2400° C., at equilibrium pressures from 0.01 to 760 mm.

Some Metallurgical Characteristics of Induction Furnaces as Determined by the Absorption of Oxygen by Molten Nickel. By F. R. Hensel and J. A. Scott. (*Preprint.* 7000 *words.*) The paper deals with the investigation of two types of coreless induction furnaces, a 60-cycle and a 5000-cycle furnace. Nickel was used as test material and the absorption of oxygen by molten nickel was chosen for determining the influence of the frequency on the metallurgical characteristics of the furnaces. Dr. Herty's aluminum method, slightly modified, was found suitable for determining the nickel oxide content in nickel. Also an electrolytic method for determining alumina in nickel was worked out. The solubility-temperature relations for nickel oxide in molten nickel were determined approximately. It was found that the solubility of nickel oxide increases with the temperature. Comparative tests on the rate of oxygen absorption made in the two furnaces showed that the absorption of nickel oxide in nickel is an increasing function of turbulence, time and temperature. The rate of oxygen pickup in the 60-cycle furnace is about three times as great as in the 5000-cycle furnace. In the 60-cycle furnace the maximum oxygen content of the melt comes close to the saturation limit. Melting under an oxygen-free atmosphere prevents oxidation in both types of furnaces.

Coal Division

Southern High-volatile Coals for Gas and Metallurgical Uses. By Howard N. Eavenson. (*Tech. Pub. No.* 489. 11,000 *words.*) This material was compiled in answer to inquiries as to the sources and reserves of the very highest grade of coals for coking, gas and metallurgical purposes. It is written from the mining and commercial, rather than from the geological, viewpoints. A general description of each of the available high-volatile seams in southern West Virginia, southwestern Virginia

and eastern Kentucky is given, but the analyses given, and the reserves computed, are only for those areas shown on the maps which yield face samples containing less than 6 per cent ash and 1 per cent sulfur.

Some Physical Characteristics of West Virginia Coals. By C. E. LAWALL and C. T. HOLLAND. (*Preprint;* also *Trans.*, vol. 101. 6500 *words.*) This paper presents the methods and results of laboratory tests on some of the physical characteristics of twenty-five of the principal coal seams in West Virginia. The properties studied are friability, crushing strengths, elasticity, and specific gravity. The tumble test is used to determine the friability. Crushing strengths and elasticity are determined on 3-in. cubes of coal. The specific gravities are determined on an air-dry basis and also after the coals had been immersed in water for ten days. Relationships between fuel ratio and friabilities, between specific gravity and ash content, and between specific gravity and fuel ratio are shown.

Some Physical Properties of Pennsylvania Anthracite and Related Materials. By J. LELAND MYER. (*Tech. Pub. No.* 482. 8000 *words.*) Values of specific heats are reported for a series of coals and other forms of carbon and for certain other materials such as slate and quartz sand for comparison. The specific heats range from 0.17 for slate to 0.31 for certain bituminous coals. Electrical resistivity of powdered samples, and variation of resistivity with temperature, pressure and moisture content, are shown for the various samples by tables and curves. The variation of resistivity with moisture content and also the rate at which moisture is lost on drying, or acquired again on exposure after drying, indicate that about 12 to 15 per cent of the moisture in coal is combined in some way, the other 85 to 88 per cent being free. Lampblack, graphite, coke, graphitic anthracite, and quartz sand show no evidence of such combined moisture. Thermal expansion values for coal are found to be rather nearer to those for wood than are those for the common forms of carbon.

Classification of the Coals of the Arkansas-Oklahoma Fields. By THOMAS A. HENDRICKS. (*Trans.*, vol. 101. 3400 *words.*) A brief description of the coals in the different districts of Arkansas and Oklahoma and their present commercial classification, urging the need for a scientific classification that shall be more nearly in accord with their physical and chemical properties and shall place them in their proper position in a general scheme of classification.

Moisture Determination for Coal Classification. By EDGAR STANSFIELD and K. C. GILBART. (*Preprint;* also *Trans.*, vol. 101. 8000 *words.*) Workers on the lower rank coals have for some time insisted that such coals, at least, should be classified on the moist-coal basis. It is a serious difficulty that the sample received by the analyst may contain free mine water or may be partly dried. A rapid, routine procedure has been worked out, and is here described, for determining the "true moisture" of the coal. This true moisture, for classification purposes, is assumed to be the minimum amount of water in fresh coal which can exert the same vapor pressure as free water at the same temperature. Tables and curves are given representing the moisture-holding characteristics of a number of Alberta coals. Methods for partial drying or air-drying of coal are outlined, as well as methods for determining residual moisture. The nature and magnitude of errors in reported values, due to oxidation and decomposition, are discussed. Two forms of apparatus for air-drying coal samples to a standard humidity, and an apparatus for the determination of moisture by distillation with xylene or toluene are described.

Condition of Water in Coals of Various Ranks. By A. W. GAUGER. (*Preprint;* also *Trans.*, vol. 101. 6000 *words.*) This paper discusses the accepted method of determining water in coal and shows that coals of all ranks do not give up all of their water even when heated in a vacuum to 110° C. for 1 hr. The abnormal vapor pressure of the moisture in coals is discussed, and some conclusions are drawn with reference

to the colloidal structure of coals of all ranks as well as to the nature of moisture in coal.

Determination of the Alkali-soluble Ulmins in Coal. By EDGAR STANSFIELD and K. C. GILBART. (*Preprint;* also *Trans.*, vol. 101. 2000 *words.*) Decaying plants form a brown, pasty mass largely soluble in alkalies. This brown matter has been termed "ulmin." Low-rank coals little removed from the peat stage, such as lignite, contain notable percentages of alkali-soluble ulmins, but the percentage decreases with coalification and is negligible in bituminous coals. The determination of these soluble ulmins is therefore a measure of coalification or rank. A procedure is described for determining the alkali-soluble ulmins in coal. The coal is fused with solid caustic potash with rigid exclusion of air, and the extracted, ulmins estimated either by precipitation and weighing, or by titration with potassium permanganate. The results are quoted for 13 Alberta coals, ranging from lignite to anthracite, and the relation of these results to the classification of coal is discussed. Insoluble ulmins can be made soluble by oxidation, but this well-known phenomenon is not treated in the paper. Attention is called, however, to a decrease in solubility when coal is first heated.

Physical and Chemical Properties of Coal in Relation to Classification. By H. F. YANCEY and K. A. JOHNSON. (*Preprint;* also *Trans.*, vol. 101. 20,000 *words.*) This report presents a correlation in graphic form of two systems of coal classification with the results of a laboratory investigation of the friability, slacking characteristics, low-temperature carbonization, and agglutinating property of approximately 100 coals from Washington and other States. The classification based on fixed carbon content and heating value is preferred to that based on carbon, hydrogen, and oxygen, because proximate analyses are more readily available, and because the range in fixed carbon content from low-rank to high-rank coals is greater than the range in total carbon content. With respect to the bases upon which the analyses are plotted, the as-received, that is, bed-moisture basis, is considered superior to the moisture-free basis, not only for presenting the relationships between the different ranks of coal from lignite to anthracite, but also for correlating the relationships between rank and the physical and chemical properties studied in this investigation—friability, slacking characteristics, and yield of products on carbonization. The use of the as-received, ash-free basis for classification purposes (1) results in less overlapping among coals of questionable rank, (2) represents more nearly the commercial product as marketed, and (3) portrays more satisfactorily the successive changes to which the coal-forming material has been subjected and the physical properties of the whole range of products resulting from these changes than does a classification based upon coal calculated free of both moisture and ash.

Status of Scientific Classification of American Coals. By W. T. THOM, JR. (*Trans.*, vol. 101. 6000 *words.*) A résumé of the work that has been done on coal classification. Tests by which group, class and type limits can be accurately and definitely fixed remain to be worked out, but sufficient progress has been made to indicate that the B.t.u. value and fixed carbon content of coals as they occur in nature will be the essential criteria on which a scientific coal classification must be based. It has been decided that coals are to be divided into four major groups of classes (anthracitic, bituminous, subbituminous and lignitic), and that classes within both the anthracitic and lignitic groups are to be recognized, while at least five and probably six subdivisions of the bituminous-subbituminous groups are to be set up. Four types of coals—woody or "humic," splinty, cannelloid, and algal or boghead—are probably to be recognized. It is probable that when the final comprehensive classification is set up it will be published with parallel statements showing the degrees of correspondence between it and other systems now in use (such as the Seyler and Parr systems) which are being currently applied to coals within restricted ranges of composition and character.

Proposed Method for Determining the Oxidation Temperature of Anthracite. By J. Leland Myer. (*Trans.*, vol. 101. 4000 *words.*) No simple definition of oxidation temperature of coal can be established on a theoretical basis, but by subjecting coals to suitable experimental conditions various oxidation characteristics can be obtained and examined for critical or significant stages of the oxidation process, and oxidation temperature itself must be defined on the basis of such experimental results. For anthracite a definite break in the slope of the temperature-resistivity curve has been found. The corresponding temperature is that at which rapid oxidation might reasonably be expected to take place. This temperature-resistivity test is simple and control of the important variables is easy. The reproducibility of results is encouraging and may enable further correlations in classification and comparative evaluation of coal.

Application of Ash Corrections to Analyses of Various Coals. By A. C. Fieldner, W. A. Selvig and F. H. Gibson. (*Preprint;* also *Trans.*, vol. 101. 5500 *words.*) The paper describes various methods of arriving at the composition and calorific value of pure coal; that is, of the coal free from its mineral matter. Such information is essential for any system of coal classification based on coal analyses. The Parr formula for "unit coal," which assumes that practically all the sulfur in coal is present as pyritic sulfur; and a modification of the Parr formula recently proposed by Fieldner and Selvig, which assumes that one-half the sulfur is in the form of pyritic sulfur, were subjected to tests on various coals by (1) float-and-sink methods, (2) computations on analyses of coals with varying ash content, from the same mine, and (3) comparison with the Stansfield and Sutherland graphic method. Although the assumption that one-half the sulfur in coal is present as pyritic sulfur is approximately correct for most coals, the tests applied and described in the paper fail to show that the modified formula has any advantages over the regular Parr formula for "unit coal." Both formulas require additional correction for carbon dioxide if carbonates are present in appreciable amounts in the coal. The graphic method of Stansfield and Sutherland gave satisfactory results when applied to a number of coals. Close agreement with the graphic method was obtained by float-and-sink separations in a solution of 1.38 sp. gr. and applying the Parr formula to the analyses of the float portions. The graphic method for arriving at the composition of pure coal has the disadvantage of the labor involved in the float-and-sink separations and the number of analyses required. For low-ash coal it is considered that sufficient accuracy for coal classification purposes can be obtained by the application of the Parr formula to the analysis of the untreated coal. For high-ash coals, however, the errors inherent in such formulas may give relatively large errors in the estimated composition of the pure coal, and for such coals it is best to reduce the mineral matter by a float-and-sink separation and then apply the Parr formula to the analysis of the float portion.

Properties of Coal Which Affect Its Use in the Ceramic Industry. By William E. Rice. (*Preprint;* also *Trans.*, vol. 101. 4000 *words.*) The paper describes briefly the purposes and methods of the burning of coal in ceramic kilns, then describes the qualities of the coal that best fulfil the requirements imposed by these purposes and methods. The ideal coal is one that produces a long smokeless flame, does not cake strongly in the fuel bed, is low in moisture, ash, and sulfur; high in calorific value, and with ash of high softening temperature. The coal should be clean and of uniform size, preferably about 2 by 4-in. egg, so that it will burn uniformly in the small furnaces of kilns. It should be hard enough to withstand shipping and handling without breaking. The ash should be low in content of sulfur and iron, because these substances in the ash that is deposited on the ware in the kiln may cause discoloration. The deviations, from the ideal qualities, of some coals that are used are discussed with reference to the conditions under which satisfactory results are secured.

Use Classification of Coal in the Portland Cement Industry. By H. P. Reid. (*Preprint;* also *Trans.*, vol. 101. 2000 *words.*) A cement plant may use fuel for (1) power generation; (2) drying raw materials and coal; and (3) burning of cement clinker in kilns. The first use is not a large one as most plants purchase power or generate it from waste heat in kiln gases. Drying, for dry-process plants only, may be done with any fuel that can furnish moderate temperature gases for removing surface moisture from raw materials. Burning of cement clinker is similar in the wet-process and dry-process plants, the wet or dry raw materials passing through a rotary kiln, counterflow with the hot gases. Work done by the heat consists of: (1) removing moisture and organic materials; (2) heating to about 1400° to 1700° to drive off CO_2 from carbonates; and (3) heating further to about 2600° to permit the chemical combinations necessary for cement clinker. Coal for kiln use must be finely pulverized, low in sulfur, and of nominal ash content. Fuel requirements may be less than 1,000,000 B.t.u. per barrel for certain dry-process plants and as high as 2,000,000 B.t.u. per barrel for certain wet-process plants, according to nature of raw materials and flame characteristics.

Recent Research on Ground Movement Effects in Coal Mines and on the Strength of Coal and Roof Supports. By George S. Rice. (*Trans.*, vol. 101. 11,000 *words.*) Description and discussion of investigations made in Germany, France, Great Britain and the United States. Tests were made in the laboratory of the bearing strength and mode of rupture of small specimens of coal and coal measure rock and of the strength and compression of artificial roof supports. Underground tests were made of unit roof pressure, movements of the roof, floor, and sides of the face working places and passageways, and also the loading and deflection of artificial supports; of the elasticity, plasticity, and flow under load of coal-in-place, when acting as a buttress for stoppings and hydraulic dams, and determination of its safe unit bearing strength, parallel and perpendicular to the coal bed; of the loads carried and deflection of different kinds of artificial roof supports in the face workings, and of their spacing and arrangement; of arrangements in longwall working of pack-walls, strip packing; also partial and complete stowing or gobbing by hand, by hydraulic means, by pneumatic agency, and by mechanical stowers; and in headings or entries and by trial of steel arches, massive concrete arching, and concrete block and brick arching. Tests of the electrical resistivity of coal were made in the laboratory and also in the face of workings to determine if possible whether there is a difference in resistivity of coal in normal condition from that in an area subject to instantaneous outbursts of gas with a view to getting warning of impending outbursts with advance of a face or heading.

Qualities of Coal and Coke Required in Nonferrous Metallurgical Industries. By Clyde E. Williams. (*Trans.*, vol. 101. 3800 *words.*) Requirements for coal used in reverberatory smelting of sulfide ores of copper are the usual ones for other uses of pulverized coal, as, for example, for power plants. Because of location, many smelters cannot choose the most desirable coal, but for economic reasons use the most accessible coal, which may be inferior. Low ash (8 per cent maximum when possible) of high fusion point and low sulfur contents are preferred. High sulfur content is not important from a metallurgical point of view, but if present as coarse pyrite might present a fire hazard and difficulties in grinding. All grades of coal from subbituminous to semibituminous in a wide range of analyses are used. Coal used in pulverized form for copper-refining furnaces must be low in ash, 8 per cent maximum being customary. Ash of high fusion point is preferred. Sulfur content must be low, a maximum of 1 per cent being preferred. In the smelting of zinc ores by the old retort process, coal or coke is used as reduction fuel. A low ash content is desired but seldom obtained. Ash of high fusion point is required for treating ores of low fusion point. A low sulfur content is preferred, 1 per cent maximum being commonly used. A minimum of 1 per cent and a maximum of 5 per cent volatile matter is

preferred. In zinc smelting by the continuous retort process, coal of low sulfur and low ash contents is required. Part of the reduction fuel must have strongly caking properties to bond briquets. Ash fusibility and content of volatile matter are not important. For the manufacture of electrodes used in the smelting of aluminum, anthracite under 2 per cent in ash is required. Petroleum coke or pitch coke is used in America, but in Europe Welsh anthracite of 1 to 4 per cent ash content is used. Requirements of coal for sintering zinc ores and desired properties of coke for lead and copper blast-furnace practice also are given.

Bituminous Coal and Scientific Research. By A. W. GAUGER. (*Min. & Met.*, May, 223. 2400 *words*.) The paper analyses the situation in Pennsylvania, to see what scientific research is needed and what can reasonably be expected of it, bearing in mind that the aims must be (1) the recovery of more nearly 100 per cent of the fuel in the ground; (2) bringing the fuel to the ultimate user in some form that will always grade as No. 1, and (3) the assurance of adequate wage to the miner, a fair profit to the operator and a reasonable cost to the consumer. It enumerates the projects on coal that are being studied in the Pennsylvania State Experiment Station and concludes that a fundamental study of Pennsylvania coals is vital to the industry.

Petroleum Division

What Is the Policy of the Mineral Industry? By C. K. LEITH. (*Trans.*, vol. 98. 2000 *words*.) Curves of consumption of minerals are apparently flattening but the curve of development is not flattening to the same extent. A study of conservation is needed, including anti-trust laws and international relations. The Mineral Inquiry, with the cooperation of other organizations, is trying to collect some of the basic facts on which to formulate such a policy. Political enactments should receive more attention from the mineral industry.

Stabilizing the Oil Business. By AMOS L. BEATY. (*Trans.*, vol. 98. 1600 *words*.) Curtailment of production of crude oil is needed; statutory enactments in which commissions and umpires administer the law, see that all producers alike curtail their operations and that one pool is not discriminated against in favor of another. The existing state laws are not perfect but the results are not bad. For the sake of investors as well as owners and operators the oil industry must not be wrecked by uncontrolled production.

Stabilizing Influences for the Petroleum Industry. By EARL OLIVER. (*Trans.*, vol. 98. 4500 *words*.) Methods of capture have led to wasteful practices in the oil industry. Lawyers assert that the state, under its police power, has the authority to regulate and prescribe methods for the extraction of oil and gas from a common pool on either of two bases: (1) to protect the public interest against waste of the products; (2) to insure a just distribution among the collective owners. They say that if engineers can devise methods of determining the relative acreage content of each landowner in a common pool within reasonable bounds it would seem to follow clearly from certain court decisions that the state has power to substitute acreage content as the standard of each owner's rights in a common pool for the old standard that he may extract from a common pool all the oil and gas he can draw out through wells drilled on his own land. This acreage content standard would help to establish proration on a sound basis in pools where proration is applied and would serve as an effective step toward promoting unit operation in pools where that method is desirable. The general opinion is that engineers know how to determine relative acreage content as set forth in the legal opinion, or can learn to do so. Decisions on matters of common interest should be determined by the collective voice of the owners in the pool. Unit operation need not be a consolidation of ownership; it need be only consolidation of operations. A proper system of unit operation would benefit owners and labor and improve social conditions.

Changing Concepts in the Petroleum Industry. By J. B. UMPLEBY. (*Min. & Met.*, May, 231; also *Trans.*, vol. 98. 3500 *words.*) The new concepts of the petroleum industry result largely from new information concerning the chemical and energy relationships of gas. The function of gas in the development and production of oil is the most fundamental concept in the industry. It permeates laws, judicial decisions, and practices. It modifies policies of exploration, development, production and marketing. It affects the public through conservation of a wasting resource; the producer through costs, reserves and stability; and the consumer through assured supply at a fair price. A proper appreciation of the function of gas makes for reserves in the ground rather than costly and wasteful storage above ground, and for sound conservation of oil. The factors limiting the maximum use of gas in the development and production of oil are divided ownership and antiquated laws. The latter are undergoing important though slow modification and it is to be expected that ways will be found to coordinate the former. Most efficient development and operation calls for legal recognition of oil and gas as belonging to the owner of the mineral rights in the particular tract in which they occur and of reservoir energy as a common attribute of the pool to be used in the maximum production of oil. It should be clearly recognized that the individual pool is the one economic unit established by nature in the oil industry.

Propositions and Corollaries in Petroleum Production. By L. C. SNIDER. (*Trans.*, vol. 98. 7500 *words.*) The peculiarities of petroleum production result, at least in large measure, from two fundamental conditions that do not apply to other industries: (1) The industry has been governed since its beginning by the law of capture; (2) until the last two or three years, the industry operated under a rapidly increasing demand for its raw product. In many other ways the business of petroleum production has been one of extremes. The interaction of the law of capture and the physical conditions that govern the flow of oil from pools has made petroleum production a business of feverish activity and haste. Probably in no other case has the old saying that "haste makes waste" been better exemplified. Investigations on the effect of gas dissolved in oil indicated that if sufficient pressure can be kept on the oil in the reservoir to prevent gas from coming out of solution the oil will flow freely to the wells for considerable distances under a very small pressure gradient. Since no complete control of gas output and pressure and of drilling can be secured so long as the lands of each property owner must be developed individually and so long as the oil belongs to the man who can capture it, Henry L. Doherty recommended that each pool should be operated as a unit. Many objections have been raised to this plan, but there has been a pronounced trend toward such operation within the past few years, and answers to the objections are being found. Although it is not suggested that all troubles of the industry will be solved by replacing the law of capture with the law of ownership in place, it is believed that the proposition that petroleum and natural gas belong to the man under whose land they lie is fundamentally more nearly correct than the proposition that they belong to the man who can capture them, and that the corollaries that may be developed to fit individual cases will be more equitable than the corollaries of frenzied haste, offset drilling, overdrilling, and so forth.

The Petroleum Products Situation. By ALBERT J. MCINTOSH. (*Trans.*, vol. 98. 4000 *words.*) Overproduction of crude oil and decreased demand for products have created a problem in the petroleum industry. The author believes that increased consumption is the real hope. He advocates a search for new uses of petroleum and its products and more energy in pushing the specialties now in use.

Economics of Proration. By JOSEPH E. POGUE. (*Preprint;* also *Trans.*, vol. 98. 3500 *words.*) Proration in the petroleum industry has come to mean a method for curtailing the production of crude petroleum by artificial effort. Curtailment plans appear to have a legitimate place in our economy, provided they are used to supple-

ment the law of supply and demand, and not to defy it. Properly conducted, they can be made to serve a valuable economic function. But if they are managed with the idea that economic tendencies can be ignored and higher prices can be attained than are justified by underlying conditions, they will prolong and intensify the period of adjustment. Proration has contributed to the building up of a huge invisible inventory, to the translation of overproduction into excess capacity, to the raising of costs, to the retardation of essential economic adjustments in the industry, and to the introduction of a political factor into the administration of the oil business. On the other hand, it has aided in the establishment of the principle of rateable takings, an important step toward unit operation, and has increased operating efficiency.

Proration in Texas, 1931. By DAVID DONOGHUE. (*Preprint;* also *Trans.*, vol. 98. 1400 *words.*) Brief mention is made of the East Texas situation, the various legal proceedings, changes in the proration program, and proration problems. The most important events of the year were the enforcing of proration in East Texas with the state militia and the taking over of the umpire system by the Railroad Commission.

Economics of Domestic Marketing. By S. A. SWENSRUD. (*Preprint;* also *Trans.*, vol. 98. 8000 *words.*) Cut-price marketing has begun to force margins down. In 1931, many major companies introduced third-grade gasoline at reduced prices with narrower margin to dealers. This was the most significant development of the trend toward lower margins. In California restoration of high margin gasoline prices by major companies resulted in decline from 82 to 67 per cent of total state sales. There was also a trend toward lower motor-oil prices, which probably will develop further. Prices of advertised motor oils have never followed the downward trend of petroleum prices in general, but now reduced demand with no reduction in supply is beginning to have an effect. Oil marketing is a fruitful field for lowering the cost of distribution because of present overdevelopment and inefficiency.

Some Influences of Foreign Demand on the Domestic Oil Situation. By E. B. SWANSON. (*Preprint;* also *Trans.*, vol. 98. 4000 *words.*) Variations in foreign demand for gasoline influence the domestic oil situation equally as much as changes in domestic demand. Foreign demand for United States gasoline has declined relatively over a long period and actually during the past year. About one-half of last year's loss may be credited to the decline in consumer demand abroad and the remaining half to the increased competition from foreign sources of supply. Fuel oil has been influenced also by increased foreign production, although the effect was noticeable earlier and has been confined principally to Latin-American markets. There also is evidence of increased foreign production of lubricating oils.

Economic Aspects of the Oil Situation. By H. J. STRUTH. (*Trans.*, vol. 98. 2800 *words.*) Among the economic factors that were responsible for the unsatisfactory conditions prevailing in the oil industry last year should be considered first of all the excessive rate at which refining facilities were operated. Undoubtedly burdensome gasoline inventories constituted the principal cause of shrinking market values and prevented materialization of accrued benefits that might otherwise have been prompted by statistical improvement in the producing branch of the industry. Operations in East Texas, before the establishment of martial law, undermined the oil industry's price structure. However, reduced production, development and refining indicate a turn toward recovery.

Domestic Petroleum Production in 1931. (*Preprint.*) The fields covered are: Kansas, by H. S. Bryant, 5000 words; Texas, except the Gulf Coast and Panhandle, by M. G. Cheney, 3000 words; Gulf Coast of Texas and Louisiana, by L. P. Teas, 3500 words; Texas Panhandle, by William E. Hubbard, 4000 words; South Arkansas, North Louisiana and Mississippi, by H. K. Shearer, 2500 words; Rocky Mountain District, by R. Clare Coffin, 4000 words; California, by V. H. Wilhelm, 7000 words;

Michigan and the "Trenton Rock" Fields, by M. G. Gulley, 4000 words; and Illinois, Southwestern Indiana and Western Kentucky, by Alfred H. Bell, 2000 words.

Printed also in *Trans.*, vol. 98, with the addition of New Mexico, by Walter B. Lang, 1200 words; Oklahoma, by T. E. Weirich, 1600 words; Eastern States, by L. G. Huntley and J. R. Wylie, Jr., 2000 words.

Foreign Petroleum Production in 1931. (*Preprint.*) The fields covered are: Russia, by R. C. Beckstrom, 3000 words; Venezuela, by Fred H. Kay, 5500 words; Trinidad, by W. J. Millard, 2800 words; and Rumania, by Ionel I: Gardescu, 2500 words.

Printed also in *Trans.*, vol. 98, with the addition of Germany, by W. A. J. M. van W. van der Gracht, 1700 words; Iraq, by B. B. Cox, 2500 words; Colombia, by O. C. Wheeler, 1300 words; Peru, by Oliver B. Hopkins, 400 words; Mexico, by Valentin R. Garfias, 2500 words; Canada, by Linn M. Farish, 2600 words.

Legal Aspects of Limitation of Oil Production to Market Demand. By ROBERT E. HARDWICKE. (*Min. & Met.*, October, 441. 4800 *words.*) That the State may regulate the production of oil and gas to prevent waste is now well settled. The decisions justify the conclusion, or at least the argument, that the right also exists to regulate production, irrespective of waste, in order to protect and adjust the correlative rights of the various owners of an oil and gas pool.

The East Texas Oil Field. By F. H. LAHEE. (*Trans.*, vol. 98. 7500 *words.*) The East Texas oil field, for the most part in Gregg and Rusk counties, Texas, was discovered by three widely separated wells, completed between September, 1930, and January, 1931. Through rapid development, the pool was definitely outlined, except at its extreme north and south ends, by September, 1931. Its total area, based on estimates made at that time, is about 92,000 acres. Estimates on average sand thickness and porosity, with an assumed extraction of 40 per cent, give the ultimate total yield of the pool as about 2,100,000,000 bbl. The oil in this pool has a gravity averaging between 39° and 40° A.P.I. It has accumulated in sands of the Woodbine formation of Upper Cretaceous age, where these sands, dipping westward from the Sabine uplift, are unconformably overlain by the Austin chalk. Owing to this relation, there is no edge water present on the east or updip side of the pool where the sands wedge out. On the western border edge water occurs below the 3320-ft. subsea contour. Depths to the pay are between 3430 and 3800 ft. Drilling is accomplished by rotary tools. The average cost of a completed well is $19,000.

Relation between Gas Energy and Oil Production. By BYRON B. BOATRIGHT. (*Trans.*, vol. 98. 3000 *words.*) The gas pressures existing in oil-producing formations may be the result of a combination of several factors. Water pressure probably has been responsible for the pressure found in most oil horizons. Gas may be absorbed in the oil, occluded or in liquefied form. Although the work that a certain amount of gas will do under subsurface conditions can be calculated with a fair degree of accuracy, it is extremely difficult, if not impossible, to evaluate mathematically the work that must be done to cause a definite oil flow in a reservoir rock. Underground conditions are changing constantly and sufficient fundamental data are hard to obtain or are nonexistent. The work on permeability by Barb and Branson [*Int. Petr. Tech.* (July, 1931) 325.] indicates a possible laboratory approach for the development of a method of determining the work that must be done to cause a predetermined flow of oil and gas from a sand under given pressure conditions. A large amount of additional information is needed, however, and until some method of calculating the energy consumption incidental to fluid movement through revervoir rocks under various flow conditions is devised, the production engineer must of necessity use the formation gas-oil ratio as the criterion for relative flow efficiencies. While this ratio gives a fairly accurate basis for comparing relative flow efficiencies in a given well, it cannot be applied quan-

titatively to a flow efficiency analysis of a pool unless reservoir pressures are also taken into consideration.

Advantages of Flowing Wells through Tubing. By HALLAN N. MARSH and BRUCE ROBINSON. (*Trans.*, vol. 98. 2000 *words.*) Analysis of experiments at Santa Fe Springs proves that wells flowing through tubing, while having a lower initial rate of production, quickly surpass in cumulative production that of comparable wells flowed through casing. Gas-oil ratio data indicate that the tubed wells will have not only a greater flowing production but a greater ultimate recovery. These favorable results with tubed wells were secured in spite of drainage by many offset wells flowed through casing. Data are presented in the form of curves showing the average performance of wells with and without tubing. These curves are largely self-explanatory and are the essence of the paper. Eight definite findings derived directly from the curves are stated. The economic advantages of securing the same or greater ultimate recovery with a lower peak rate of production are pointed out, and the advantages of requiring the use of tubing as a basis for curtailment are suggested.

Flowing Wells with Small Tubing. By R. R. HAWKINS. (*Trans.*, vol. 98. 2500 *words.*) Wells no longer able to flow through 2-in. tubing can be made to produce through smaller tubing if conditions within the well are carefully studied and correctly designed strings are installed. The paper discusses determination of exponential flow, slippage and friction, velocity control, volume of gas a well will produce at the pressure required for flowing, and the design of the string.

Selection and Use of Screened Pipe. By CLIFFORD S. WILSON. (*Tech. Pub. No.* 490. 10,000 *words.*) Screened pipe is designed to restrain the fine, free-running sand of the production zone from entering the well. Conditions requiring consideration in the selection and use of the pipe are discussed. Harmful effects of sand production are pointed out and details of the cutting out of liners are outlined. Advantages and disadvantages of screened pipe compared with perforated pipe are summed up as to cost, rate of production, time required to set, action with sand, caving, collapsibility and pump troubles. The necessity of proper sampling and screen analysis of the samples is stressed. For the selection of the proper mesh of screen to be used in any given case a mathematical method, based upon screen analysis, is proposed. The required steps in the calculation are set forth in detail and the method is illustrated with the citation of an actual computation carried out by the author in field practice.

Control of Gas-oil Ratios in the Yates Field, Pecos County, Texas. By M. ALBERTSON AND W. A. SCHAEFFER, JR. (*Trans.*, vol. 98. 7500 *words.*) This paper presents a method of applying the principles and theory of gas-lift action to naturally flowing wells. Practically all flowing oil wells depend on gas-lift principles. In actual practice engineering development of the principles has proceeded but a short distance into this field of endeavor. The writers doubt that 5 per cent of its ultimate commercial application has been adequately developed in any oil pool of the United States. The method is perhaps, with few exceptions, only in the ox-cart stage of development. Two sets of factors determine gas-oil ratio in flowing wells; one group is in the flow column, the other is in the reservoir. By proper design of flow columns much gas can be saved. Other methods, such as advantageous selection of well locations on the structure and most favorable rate of withdrawal from the pool as a whole, as well as numerous other features which cannot be discussed in a paper of this length, will increase oil recovery, decrease production costs and conserve gas. Use of tapered-tubing flow columns in the wells of one lease of the Yates field has reduced gas wastage by more than 85 per cent.

Recent Development and Use of Bottom-hole Choking. By J. S. ROSS. (*Trans.*, vol. 98. 4500 *words.*) This paper gives a discussion of the various applications of bottom-hole choking in the operation of oil and gas wells, based upon recent experience with a removable choke operating under different flow conditions in many Mid-

Continent fields. It is shown that bottom-hole choking, while not applicable to all flow conditions, can often be applied successfully in wells in which an inefficient low flow velocity is due to a shortage of available reservoir gas or to excessive surface choking, the function of the choke in this instance being to accomplish an increase in the velocity of the rising column of gas and fluid, particularly in the lower part of the flow string. Often bottom-hole coking brings about steadier flow and prolonged flowing life and in some instances decreased gas-oil ratios and increased rates of production. The use of bottom-hole chokes presents a new application in connection with the operation of gas wells, since their installation in siphons permits a more efficient water disposal and practically disposes of freezing difficulties. The type of bottom-hole choke used is a slip-packer arrangement, which is lowered and pulled under pressure on a steel measuring line and which is set at any desired depth without disturbing the tubing string.

Reservoir Pressures in the Hobbs Field, New Mexico. By R. S. Christie. (*Trans.*, vol. 98. 4000 *words.*) The reservoir at Hobbs is in limestone, therefore producing rates or conditions may be especially sensitive to changes in pressures. The paper reports measurements of closed-in pressures taken with an Amerada pressure gage and interprets them in terms of production. Several tests were run on flowing wells to determine the drop in pressures at different flows. The paper suggests that a similar method could be used in establishing potentials.

Determination and Application of Depth Pressures in the Yates Field. By Dale Nix. (*Trans.*, vol. 98. 5000 *words.*) The "compressor," the "volume chamber" and the "depth pressure recorder" methods are used to record depth pressures in the Yates field. They are reasonably accurate and generally applicable. Approximately 400 depth pressures have been determined and applied to production and proration problems, including determination of well potentials, adjusting well potentials for proration purposes, determination of fluid column weights and pressure gradients, determination of favorable flowing conditions, and study of water encroachment. They suggest other uses for depth pressures. The author acknowledges the use of data compiled by the Yates Engineering Committee.

Effects of Rate of Production and Producing Equipment upon Gas-oil Ratios. By J. T. Hayward. (*Trans.*, vol. 98. 3500 *words.*) Where the rate of production is fixed, it is impossible to control or alter the gas-oil ratio at a well by means of the equipment above the sand. Changes in the lifting efficiency, or in the oil-gas slippage, do not affect the gas-oil ratio unless accompanied by a change in the rate of oil production. Where there is no free gas at the pressure prevailing at the botton of the hole, the gas-oil ratio varies with the rate of production, and it is possible, therefore, to improve the ratio by selecting a favorable rate of production. If this favorable rate of production can be obtained, the equipment will make no difference in the ratio. It is not argued that tapered strings, bottom-hole chokes, etc., have not their uses. They may, for instance, enable the natural flowing life of a well to be extended, or lower the input-gas requirements of gas-lift wells, reducing the lifting costs. The position of the lower end of the flow string, when below the top of the sand, may affect the gas-oil ratio, but this is the only point in connection with the equipment by which any control over the gas-oil ratio can be obtained. Where the gas is above the oil, as in most cases, lowering the tubing below the gas sand will tend to improve the gas-oil ratio. The gas-oil ratio of a pool, considered as a whole, during the early part of its life, may, if there is free gas, be varied over wide limits by selecting the position on the structure from which the production is withdrawn, even when the rate of production is determined by other considerations, but the ratio at any individual well, with a predetermined rate of production, is beyond control.

Flow of Air Gas Through Porous Media. By Joseph Chalmers, D. B. Taliaferro, Jr. and E. L. Rawlins. (*Preprint;* also *Trans.*, vol. 98. 9000 *words.*)

This paper gives a preliminary report of the findings of the U. S. Bureau of Mines resulting from an investigation of the flow relationships governing fluid migration through sands. A series of over 60 tests was conducted covering the flow of air and gas through different grades of unconsolidated sands and other porous media with pressures ranging from 400 lb. per square inch to atmospheric. Plotted data show that the pressure gradient in the flow of gases is a linear function of distance when the squares of the pressures are used. This relationship and other plotted relationships based upon the difference of the squares of the pressures as a function of the rate of flow lead to the derivation of an empirical equation, which seems to fulfil the requirements of the data to a high degree of accuracy. The magnitude of the constants are indicated to be a tangible means of determining and studying the relationships of such factors as porosity, grain diameter, and shape of grain to the permeability of a sand. Mean effective pore diameter is suggested as having greater significance in flow relationships than a permeability factor. A plot is also given showing the relationship between friction factor and the Reynold's criterion. The relationships for rectilinear flow through sands are generally similar to the flow relationships for pipe lines.

A Method for Computing Pressure Drop in the Pipe of Flowing Oil Wells. By K. B. NOWELS. (*Preprint;* also *Trans.*, vol. 98. 16,000 *words.*) An adaptation of the Fanning formula is thought to include all of the variables involved in problems of vertical flow and in the form $P = 0.323fLSV^2/D$ to be satisfactory in computing the pressure drop in pipes of flowing oil wells, provided satisfactory values for "turbulence factors" DWV/U and friction factors f are first secured from data in the field.

A method is explained for utilizing the information given by gas-lift installations and bottom-hole pressure gages in establishing a set of curves for various pipe sizes, which will give the desired values of DWV/U and their definite corresponding values of the coefficient f. With these values and those of the other variables known, they may be substituted in the adopted Fanning formula in solving for pressure drop P.

A graphic method for arriving at values for U, the absolute viscosity of gas-oil mixtures for use in the expression, DWV/U, is presented. Nomographic charts are also presented to facilitate the computation of all values in the Fanning formula.

Experimental Study of Pressure Conditions within the Oil Reservoir Rock in the Vicinity of a High-pressure Producing Well. By L. C. UREN AND E. J. BRADSHAW. (*Trans.*, vol. 98. 11,000 *words.*) This paper presents the results of a program of research designed to disclose the conditions controlling flow of oil and gas through its reservoir sands in the immediate vicinity of the wall of a producing well. Particular attention has been given to the factors that influence the pressure gradient and the rate of flow. The apparatus employed is described. The data thus far secured would appear to justify the following tentative conclusions: (1) There is a mutual relationship between the pressure gradient, the field pressure, the degree of gas saturation and the viscosity of the oil and the sand permeability, which determines the rate of flow of oil through its reservoir sands; (2) The form of the pressure gradient is determined, for a given sand body and a given oil, by the rate of flow, which in turn depends upon the viscosity of the oil, the field pressure and the proportions of gas to oil. (3) A large part of the pressure loss in moving oil through the reservoir sand to a well outlet occurs in the immediate vicinity of the wall of the well. (4) The rate of oil production varies with the maximum field pressure operative, the equation of production probably taking the general form $Q = KP^n$ in which the value of the exponent of P apparently increases with the rate of flow. (5) Two essentially different processes of expulsion control the rate of production in different periods of the history of an oil well. (6) Gas-bubble resistance to the expulsion of oil from a high-pressure producing sand is of small importance in comparison with the expulsive forces operative, and the state of the gas present in the oil does not materially influence the rate of oil production. (7) Where, in flowing through a reservoir sand to a well outlet, by-passing of

the oil by gas is not excessive, the absolute gas-oil ratio (*i. e.*, the ratio of the volume of free gas at atmospheric pressure to that of the oil) remains approximately constant; but the apparent gas-oil ratio (*i. e.*, the ratio of the volume of free gas at the existing pressure to that of the oil) rapidly increases as the wall of the well is approached. Multiplication of the number of gas bubbles occupying the flow channels near the wall of the well is due to release of gas from solution in the oil as pressure is diminished, to expansion of gas and partition of large gas bubbles, and to restriction in the flow cross-section. (8) The apparatus and methods employed in this research afford a means, not hitherto possible, of experimentally studying the effect of the several factors controlling drainage of oil and gas from their reservoir sands. The effect of back-pressure and diameter of the well on drainage efficiency may be studied to advantage. A possible experimental solution for the problem of determining the drainage radius of a well is suggested.

Some Experiments on the Behavior of Natural Gas in an Oil-sand Reservoir. By IONEL I. GARDESCU. (*Preprint;* also *Trans.*, vol. 98. 3000 *words.*) A description of three experiments showing that by creating different distributions of gas masses in an oil-sand reservoir it is possible to secure different rates of evolution of gas and consequently different rates of production of oil. The differences in distribution of gas masses were accomplished in the experimental work by causing variations in pressures and by the application of shock, both of which may be applied to a natural oil reservoir.

Flow of Drilling Mud. By H. N. HERRICK. (*Preprint;* also *Trans.*, vol. 98. 5000 *words.*) Computations regarding flow of drilling mud through pipe and accessory equipment can be made with sufficient accuracy for practical purposes by assuming that the mud is a plastic solid, which has a definite shearing strength. The pressure required to force mud through pipe is calculated as the sum of the pressure required to start and maintain shear, constant for all flow rates, and a friction loss depending on the flow rate as for any liquid of 48 seconds Saybolt viscosity (9.2 centiposes). The shearing strength (yield point) of Kettleman, Calif., muds is approximately 0.02 (weight of mud in lb. per cu. ft.—65), in lb. per sq. ft. The viscosity varies only slightly from the average value above stated. Since the flow rate of mud is not directly proportional to pressure applied to it, a single viscosity determination does not suffice to define its properties for flow calculations. Several tests at different flow rates are required, as with greases and paints. The shearing strength of drilling mud affects settling of sand grains suspended in it, since there is a definite limit of grain size for each mud below which the sand particles do not have weight enough to shear the mud, and so cannot be separated from the clay by settling.

A Borehole Camera. By BELA LOW AND SHERWIN F. KELLY. (*Min. & Met.*, February, 81. 2200 *words.*) The best substitute for putting a periscope down a hole is the taking of photographs of its interior at points regarding which information is desired. An apparatus for this purpose, constructed in Holland, has been put into practical use by the Geological Survey of The Netherlands in searching for water. The instrument is described, with some examples of its work, and the authors suggest that it might be adapted to serve a variety of useful purposes in American oil fields.

Prospecting for Natural Gas in New York State. By FRANK BREWSTER, PAUL D. TORREY AND JOHN A. THOMPSON. (*Min. & Met.*, July, 316. 3000 *words.*) Prospecting by geophysical methods was begun in the Finger Lake region of New York in November, 1931, and the mapping has been followed by a triangular drilling development plan. The Wayne-Dundee fields are still in the period of flush production.

Nonmetallic Minerals

The Mineral Wool Industry in Indiana. By W. N. LOGAN. (*Preprint.* 4000 *words.*) Raw materials suitable for the manufacture of mineral wool are found in

formations of Mississippian age in southern Indiana and in formations of Silurian age in northern Indiana. The larger part of the product is from the latter area. The selection of quarry site with reference to position of transportation lines, thickness of overburden, and water supplies is of importance. The quarrying of the raw materials involves the selection of areas containing wool rock and flux rock. Methods of manufacture, which include the charging of wool rock, flux rock, and fuel into the cupolas, require careful attention. The temperature of melt necessary to produce the fiber through the propelling action of steam under pressure is important to prevent waste of fuel and product. The physical and chemical properties of wool-rock materials and of mineral wool are delineated. The composition of typical samples of wool rock and the functions of the various compounds found in the wool are given. Fabrication of mineral wool into its various commercial forms is also discussed.

Uniform Cost Accounting in the Crushed Stone Industry. By WILLIAM E. HILLIARD. (*Preprint.* 4000 *words.*) In manufacturing, the management must know the exact cost of the units of production and the different factors which make up that cost. Many trade associations have planned uniform cost accounting systems. The National Crushed Stone Association has recently developed a uniform cost accounting system, available for its members. The quarry, plant and equipment require labor and material for operating and also for maintenance and repairs. Funds for replacing by means of insurance and depreciation reserves are necessary. On that basis, a group of accounts is constructed as follows: (0) operating labor, (1) operating material, (2) repair labor, (3) repair material, (4) power, (5) sundry expense, (6) overhead, (7) charges from reserves, (8) special charges. The general headings may be subdivided, then each one of the subdivisions may be split up into the eight accounts listed above. By using this system, a producer can check one year against another and learn just where there are possibilities of cost reduction. This system also helps in the elimination of waste of labor, material and equipment. Another advantage is the opportunity to exchange cost data among different members of the Association for their mutual benefit.

The Geology and Economics of Tin Mining in Cornwall, England. By ERNEST R. LILLEY. (*Tech. Pub. No.* 479. 6000 *words.*) Tin has been obtained in Cornwall for at least 3000 years. Lode mining started in the sixteenth century. During the nineteenth century the area was worked with great profit for both tin and copper. The copper yield is no longer important because of competition from other areas and decrease in copper content with depth. Similarly, although lead and zinc were worked during the last century, they are of little importance today.

The lodes having economic value generally strike east to west and are found both in the granite masses from Bodmin Moor to Land's End and in the adjacent metamorphosed sediments. The regularity of the vertical distribution of metals in the lodes is worthy of special mention, the deeper mines (from a geological viewpoint) producing tin as the only metal of value, whereas above this zone in ascending order occur zones of tin mixed with wolfram, sulfides of copper, sulfides of zinc and lead with silver, sulfides of antimony, and carbonates of iron and manganese.

The metal content of the ore has always been comparatively low, even during those periods when the area was worked with much profit. During recent years it has been difficult to operate the deposits with profit because of the milling problems arising from the extreme fineness of the cassiterite in the deeper portions of the lodes. Additional difficulty arises from excessive quantities of water in the old workings above and connected to the present workings. Despite these problems and the competition from richer ores in other countries several conservatively organized and managed companies have made a good record during the last decade. In general, however, it appears that the tin mining industry in Cornwall has made little progress in applying the principles of modern management, engineering, research, organization, and financing used successfully in other mining areas.

Utilization of Coal-mine Waste in Concrete. By H. HERBERT HUGHES. (*Min. & Met.*, November, 488. 5000 *words.*) Utilization of coal-mine waste as a source of raw material for concrete aggregate must not be misconstrued as a "cure-all" for the coal industry, but for a few favorably located mines it may aid in profitable disposal of waste. Several specially prepared lightweight concrete aggregates are now available either commercially or experimentally. Cinders have been used in concrete ever since the advent of portland cement, but more recently several burned shale or clay aggregates have been introduced. Haydite, Cel-Sealed Aggregate, and Lytag, the most important members of this group, can be made from raw materials comparable to mine waste. The Lehigh Navigation Coal Co. has conducted extensive experimental work with breaker waste to create a satisfactory aggregate. All concrete can be grouped into two general divisions: that which is poured at the job, and that which is used in precast concrete products. The lightweight aggregates now available are used in both types of work in about equal proportions. Leading construction engineers agree that the potential markets for lightweight concrete have scarcely been touched. For a favorably located mine, the possibilities of using mine-mouth waste in concrete certainly are worth investigating.

The Kasai Diamond Fields of the Belgian Congo. By A. E. BRUGGER. (*Min. & Met.*, August, 357. 2200 *words.*) The development of a technique for the location and evaluation of the widely scattered deposits has been a process of gradual evolution over several years. Early methods were necessarily hand, but now transportable pan plants operated by electricity or steam, varying in capacity from 3 to 7.5 cu. m. per hour, are widely used. Reserves have been developed for a long period of profitable operation. The diamonds are of fine gem quality except those from the Bushimaie field, but are rather small. They average from 12 to 15 stones to the carat, with a few above 5 carats.

Placer Diamond Mining in Brazil. By W. B. STANLEY AND BURTON E. ANDERSON. (*Min & Met.*, July, 325. 1200 *words.*) A short description of the method of mining black diamonds in alluvial deposits in the prehistoric bed of the Paraguassu River in the State of Bahia.

Mining Geology

Occurrence of Lead-zinc Ores in Dolomitic Limestones in Northern Mexico. By M. W. HAYWARD and W. H. TRIPLETT. (*Tech. Pub. No.* 442. 15,000 *words.*) The object of this paper is to record and tabulate the data and field observations obtained by the writers and their associates during 10 years of intensive study of the lead-zinc deposits in the cretaceous limestone areas of northern Mexico. A large number of samples were taken, first, to prove the sharp line of contact between dolomite and limestone indicating the marine origin of the dolomite, and, second, by taking samples near to and remote from ore bodies to determine whether or not there had been introduction of magnesia during the period of mineralization. The mining districts described are Sierra Mojada, and Higueras, in the State of Coahuila; Minas Viejas, Mitra Mountain, and El Diente, in the State of Nuevo Leon; Ojuela, State of Durango, and Santa Eulalia, State of Chihuahua. At Sierra Mojada, Minas Viejas and Higueras, it is pointed out that the orebodies occur almost exclusively in dolomitic beds along the axes of anticlines, or on flanks of anticlines along control fissures. At Ojuela and Mitra Mountain, the orebodies occur in dolomitic horizons, and, in some cases, immediately underlying the dolomite, indicating possible damming of the solutions by overlying impervious beds. Four possible explanations are suggested as to why the dolomitic beds are more favorable for ore deposition and replacement than the adjacent limestone: (1) mechanical preparation of the ground for ore deposition; (2) porosity of the dolomite beds; (3) selective chemical action of the mineralizers on

the dolomitic beds; (4) damming influence of the dense thin-bedded dolomites. The writers are inclined to believe that the first two suggestions best explain the fact that the dolomitic beds are more favorable.

Geology of the Molybdenite Deposit at Climax, Colorado, and Other Deposits Producing Molybdenite. By John W. Vanderwilt. (*Preprint.* 4000 *words.*) The world's production of molybdenum has increased five-fold, compared with pre-war times, since 1925, and is now about 4,000,000 lb. per annum; 75 to 85 per cent of this increase has come from the Climax deposit owned and operated by the Climax Molybdenum Co. At Climax over 84,000,000 tons of 0.84 per cent molybdenite ore representing about 760,000,000 lb. of molybdenum has been developed and the company is equipped to double, if necessary, its present production of nearly 3,000,000 lb. per annum. The Climax deposit as described by Butler and Vanderwilt is of Tertiary age and has the form of a pipe or stock of silicified pre-Cambrian granite which enlarges downward. The molybdenite is fine grained and occurs in numerous quartz and orthoclase veinlets in a zone around a nearly barren core of quartz. The deposit is not a pegmatite as some have assumed. Questa, Taos County, New Mexico, and Knaben, Norway, are the only other important producers of molybdenum. At Questa the molybdenite occurs as a system of irregular and discontinuous veins and at Knaben the molybdenite is found disseminated in granite and in a silicified shear zone which contains 0.3 to 0.5 per cent of the sulfide. The ore reserves of these two deposits are not known but there is nothing to indicate that they are comparable to those of the Climax deposit. It is certain that unless important new discoveries are made in foreign countries the United States will continue as the largest producer of molybdenite in the world.

Geology of the Robinson (Ely) Mining District, in Nevada. By E. N. Pennebaker. (*Min. & Met.*, April, 163. 6500 *words.*) An outline of the geology is given in which it is emphasized how old fold and fault structures have helped to guide the intrusive igneous rocks into place. Two types of monzonite porphyry intrusives, closely spaced as regards their times of injection, are recognized. Bodies of the older type are of great economic importance because they have been severely fractured and impregnated by copper-bearing minerals to form extensive bodies of low-grade ore. Considerable importance is placed on the long-continued action of faults. It is believed that these breaks persisted in their movement during a long period of geologic time. The copper-bearing ground is arranged with respect to a structural pattern that was initiated by folding and faulting before intrusion and mineralization took place. Certain portions of the master fault zones are designated as channelways up which magma welled, and later these same zones served as conduits for the passage of copper-bearing solutions. The fracturing attendant upon recurring, post-porphyry fault movements formed open spaces in the rocks into which solutions migrated and deposited their copper. It is pointed out that many of the major ore-bodies are found beneath moderately flat fault "roof structures." Erosion has not been sufficient throughout the district to expose all of the ore-bearing ground, and the bulk of the porphyry ore now developed is primary in origin. Secondary enrichment, so necessary to originate ore in many of the so-called porphyry copper camps, has not been active in forming what has recently been considered ore throughout much of this district. This demands a prospecting procedure independent of surface copper showings and of evidence commonly left in leached outcrops above deposits due to secondary enrichment. The procedure evolved consists in prospecting critical structural features in the intrusive zone and in the flanking sediments.

Radium and Silver at Great Bear Lake. By Hugh S. Spence. (*Min. & Met.*, March, 147. 5400 *words.*) The pitchblende occurrences recently discovered in the Echo Bay region, Great Bear Lake, are believed to be the most extensive and important deposits of this mineral known. Two veins have been proved over a total

combined length of over 2000 ft., and are believed to converge into a strong break upon which pitchblende has been found at several points, over a distance of two miles. Samples from different openings on these veins have shown uranium oxide contents ranging from 30 to 62 per cent, equivalent to from 78 to 161 mg. of radium per ton. or 1 gram of radium in about 13 and 6½ tons of ore, respectively. On certain sections of the veins, the pitchblende is associated with extremely rich native silver ore. The region in which the discoveries have been made is exceedingly favorable for mineralization, owing to the extensive fracturing and shearing that have occurred, so further important discoveries may be expected. As the region is almost entirely devoid of overburden of any kind, except along the breaks, the surface is easily prospected. Although the region is remote, the airplane furnishes a ready means of access. Transportation difficulties can be met by improvements along the water route to the region via Great Bear River.

Prospects for Future Gold Supply. By GEORGE E. COLLINS. (*Min. & Met.*, February, 77. 4000 *words.*) With the world's alluvials virtually exhausted, and with little probability that important placers will hereafter be found, it is unlikely that, after known deposits are exhausted, the present rate of production can be maintained, much less increased in proportion to the world's requirements, if the gold standard is maintained in its present form. Even if the unexpected should happen, and a new Rand be discovered tomorrow, it would require 20 years of effort to bring it to maximum production. The present rate of supply from known deposits cannot be maintained so long.

Economic Notes on Steelmaking Alloys. By PAUL M. TYLER. (*Min. & Met.*, May, 225. 4500 *words.*) Describes in relatively nontechnical terms the alloy steel industry and studies its relations to national progress. Sources of the alloys are listed, with statistics as to the amounts used in ferroalloys.

The Place of Government, State and Federal, in Rationalizing Mineral Production. By C. K. LEITH. (*Min. & Met.*, October, 453. 7500 *words.*) Collective control of the mineral industry, both public and private, has been rising like a tide the world over, with corresponding narrowing of the sphere of individual activity and responsibility. This trend has been world wide, without regard to kinds of government and without regard to peace and war. It finds its origin in the hugely augmented scale of mineral production in the last few decades, in the fact that mineral reserves on a scale adequate to meet the new demands are very unequally distributed among nations, requiring large international movements of minerals, in the public recognition of the fact that minerals are national assets which are irreplaceable, in the efforts of nations to make themselves self-sustaining in regard to minerals, in the problem of curbing recent surplus production, and, for some countries, like Russia, in the ascendency of new political philosophies. So rapid has been the growth of public control that there is now a tendency in some quarters to go much farther in this direction as a panacea for economic troubles. Reasons are cited for resisting this tendency, and particularly for opposing the creation of new federal commissions of the kind that have been proposed for the coal and oil industries. Such commissions cannot take over the police and taxing powers of the states, or the tariff and treaty-making powers of the federal government. Even if it were possible to create public bodies with powers really wide enough to control the industry, it would be difficult to make them strictly nonpolitical; they would tend toward freezing existing interests and practices; they would hinder the many rapid adjustments inherent in diverse private control; they would tend to minimize the working of the law of survival of the fittest by which the world has so far progressed, and to substitute an untested system whereby the weak and marginal operations are likely, for political reasons, to be carried along. A more promising field of effort seems to be in the proper correlation of the public powers already existing, in order to eliminate contradictions between

the applications of police and taxing powers, or between these powers and the tariff and treaty-making powers. There is opportunity for the profitable extension of informal conferences among the states, illustrated by the activities of the Oil States Advisory Board, and among the mineral industries themselves. Modifications of the anti-trust laws in the interest of conservation also seem to be required. In short, instead of additional public bodies to control mineral resources, there is need of the formulation of a unified national mineral policy, which does not now exist, as a means of coordinating and directing the public powers already in operation.

Mining Administration

Utah Copper Plan for Rotating Employment. By J. G. HADLEY. (*Min. & Met.*, May, 229. 1700 *words.*) Since Jan. 30, 1930, the Utah Copper Co. has had in effect a plan of rotating employment which has permitted an increase of 70 per cent over the number of employees required for full-time work, and consequently has prevented unemployment to that extent. The plan was based entirely on the equitable distribution of shifts. Those in the higher wage bracket work approximately 15 days per month and the number of days per man is graduated upward as the day pay rate decreases, until the lower wage bracket is reached, the employees in this group being employed approximately 22 days per month.

Labor Conditions in Katanga. By THOMAS S. CARNAHAN. (*Min. & Met.*, July, 309. 5000 *words.*) Describing the feeding, clothing, housing and medical care of the native labor employed by the Union Minière du Haut Katanga and the conditions of work and remuneration of the European employees.

Uniform Cost Accounting in the Crushed Stone Industry. *See* NONMETALLIC MINERALS.

Geophysical Prospecting

Choice of Geophysical Methods in Prospecting for Oil Deposits. By E. DE GOLYER. (*Trans.*, vol. 97. 5500 *words.*) Geophysical methods in oil finding are (1) magnetic, (2) gravity, (3) electrical, (4) sonic or seismic. The author believes that in the present state of development of the geophysical methods for areas where the occurrence of oil pools is controlled by normal folding the results obtained from seismic surveys, reflection method, are more definite and of greater value than any other type of geophysical information obtainable. The writer's second choice for all areas, and first choice for areas in which the seismic methods are not usable, is the gravimetric method, generally a torsion balance survey. The performance of electrical methods in oil fields has not been impressive. The results of magnetic surveys are not susceptible of interpretations sufficiently definite in terms of geologic structure to give them any considerable value to the oil prospector, except in few and very special cases. The most important part of all geophysical work is the proper interpretation in geologic terms of the physical data secured from field observations.

Theory and Experiments Concerning a New Compensated Magnetometer System. By C. A. HEILAND and W. E. PUGH. (*Tech. Pub. No.* 483. 20,000 *words.*) The theory of the temperature effect on the new system is discussed in two parts. First the magnetic effect is considered; second, the combined mechanical and magnetic effects are derived. The temperature coefficient of the scale value is found to be negligible. The T. C. of the reading is dependent primarily on the magnetic latitude; that is, the T. C. increases with a decrease in latitude. The change is not great; in fact, a system adjusted for Golden, Colo., may be used without readjustment practically throughout the United States. There may be an influence of the scale value on the T. C., providing a change in the position of the scale value screw changes the lateral position of the center of gravity. The theory shows that, if the T. C. is not

determined near the reading 20, the deflection of the system enters as it changes the apparent Z; this influence is small. The experimental work required preliminary investigations such as the determination of the magnetic moment of the system and its temperature coefficient, and determination of time lag. The diurnal variation was recorded simultaneously. The same temperature gradient was used in all T. C. determinations to increase the accuracy of the observations. Temperature curves were obtained for different latitudes varying between vertical intensities near the magnetic equator and intensities exceeding Z at the poles and, finally, for different scale values. The results are shown graphically, and are in almost perfect accord with the theoretical deductions. Computations of T. C. based upon the theory have been checked by the experiments with an accuracy of $\pm$ 0.3 gammas. The new magnetic system gives reliable results for a great variety of latitude, scale value and temperature conditions. The derivation of the complete theory makes it possible to determine accurately the influence of any changes in the above factors upon the T. C., and to calculate the change in the T. C. for any changes in the distribution of the masses of a magnetic system.

Geophysical Examination of Meteor Crater, Arizona. By J. J. JAKOSKY, C. H. WILSON and J. W. DALY. (*Trans.*, vol. 97. 16,000 *words.*) Geological study indicates the presence of a foreign material in the southern portion of the crater. Indicated position of this material corresponds with the electrically conductive zone and the magnetic anomalies. The electrical survey gives indications of the presence of an area of higher conductivity in the southwest quadrant of the crater, between the center and the rim, the main mass of which lies at an effective depth of approximately 750 ft. A careful study of the original and altered materials found in the area indicates that this zone of higher conductivity is not due entirely to fill material or structural conditions. The conclusions are that this area contains material of metallic character. The material is not in the form of a sphere, but probably a fragmental zone having its greatest length in a general southwest direction. The magnetic studies indicate the presence of an area containing magnetic material in the southwest portion of the crater. This material starts at depths of approximately 200 ft. and continues downward, probably concentrating with depth. The geological evidence and the electrical and magnetic indications individually would be classed as fair or moderate effects of a buried mass. The general agreement as regards plan location, depth and other factors gives sufficient added strength to the results to warrant further explorations to the extent of churn-drill holes.

Effect of Impregnating Waters on Electrical Conductivity of Soils and Rocks. By KARL SUNDBERG. (*Trans.*, vol. 97. 11,000 *words.*) This paper is a study of the factors which mainly determine the electrical conductivity of soils and rocks; that is, the conductivity and content of water. It is of considerable practical importance to study the electrical conductivity of soils and rocks, because different strata may be identified by their electrical conductivity. Consequently both stratigraphic and structural studies may be carried out by electrical methods.

Examples are given of direct determination of electrical conductivity of rocks by different methods. It is generally difficult, or impossible, to determine the electrical conductivity directly, however, and an indirect method to study the conductivity of rocks has, therefore, been developed. The specific electrical resistance of igneous rocks and dense insoluble rocks generally is very high, the magnitude being 10^5 ohm-centimeters. The specific electrical resistance of young sediments varies tremendously and is generally low; values as low as 100 ohm-centimeters and less occur.

Electrical Coring: a Method of Determining Bottom-hole Data by Electrical Measurements. By C. and M. SCHLUMBERGER and E. D. LEONARDON. (*Tech. Pub. No.* 462. 16,700 *words.*) A technique has been evolved for the examination of

rock formations *in situ* through the observation of some of their electrical characteristics. Appropriate electrical contacts and instruments are lowered into the drill-hole and are connected with a measuring apparatus at the surface of the ground. It thus becomes possible to measure the electrical resistivity of the rocks and of the borehole fluids, the temperature of the rocks, their porosity, the direction and dip of the strata, and the inclination and direction of the hole. Through the observations of electrical resistivity, stratigraphical correlations can be made, and when combined with porosity determinations, data are obtained as to the location of oil-bearing horizons and their probable productivity. Studies of the resistivity of mud-laden fluids in boreholes indicate the points at which water is flowing into the hole. The apparatus for carrying out this work is described, as well as new devices for measuring rock temperatures, dip and strike of strata, and the deviation of boreholes from the vertical. The experience of the authors indicates that considerable savings in the cost of oil exploration can be effected by these means, and that more bottom-hole data become available as a result of employing the techniques described.

The Location and Study of Pipe Line Corrosion by Surface Electrical Measurements. By C. and M. SCHLUMBERGER and E. G. LEONARDON. (*Tech. Pub. No.* 476. 10,800 *words.*) A review of previous experimental work on corrosion is given, showing the principal laws which seem to condition the corrosion of buried metallic masses, and in particular the relation between corrosion, the resistivity of the soil, and the existence of long-line currents flowing in pipe lines. Techniques usually employed heretofore in studying these factors required excavation to make electrical contact with, or near, the pipe line being investigated, but the authors and their associates have developed a method of observation which permits an electrical study to be made at the surface of the ground of these various phenomena involved in corrosion. They show how it is possible to locate zones of corrosion, even in the difficult case of a pipe line passing through urban areas, with descriptions of the apparatus and techniques employed.

Applying the Megger Ground Tester in Electrical Exploration. By BELA LOW, SHERWIN F. KELLY AND WILLIAM B. CREAGMILE. (*Trans.*, vol. 97. 5000 *words.*) During the years of development of geophysical prospecting, the theory and technique of the electrical methods were not generally made public, with the result that only certain companies were equipped to carry out electrical surveys. The discovery that the Megger Ground Tester is applicable in geophysical exploration, however, opens up the possibility that geologists and engineers without special training in geophysics may be able to carry out simple electrical reconnaissance surveys. The need for special training in planning and interpreting the surveys nevertheless remains indispensable. With this idea in mind, a description of the Megger is given, including the wiring diagram of the instrument and a discussion of the principles upon which its operation is based. Brief consideration is given to the fundamental ideas underlying the determination by electrical means of the depth to given geological formations. The technique of drawing up a map of the electrical resistivities of geological formations over a considerable extent is also described briefly. To illustrate these two procedures, some laboratory experiments are quoted, in which a Megger was used in conjunction with a tank of water and some resistant and conductive materials, to simulate field conditions. Actual field experiments are described in which the depth to bedrock was determined at two bridge sites in Missouri. Finally, a brief summary is presented of the possible fields in which the instrument could be employed profitably, including the mapping of subsurface contours, determining the depth of overburden, tracing given geological formations, and investigating the simpler problems of ore occurrence.

Interpretation of Resistivity Measurements. By G. F. TAGG. (*Tech. Pub. No.* 477. 5500 *words.*) The paper gives a method of determining mathematically the

depth of a single horizontal stratum, based on the method described by Wenner for measuring earth resistivity. In making a survey, the central point of the electrode system is kept fixed and readings of a measured resistance taken for various values of electrode spacing in a straight line. These data give resistivity ratios for any spacing of the electrodes. By use of the curves shown in the paper it is possible to compute the depth of the planes of separation between the strata. Results of surveys, using a Megger ground tester as a convenient means for determining resistances, are given to illustrate the method.

Results of Earth Resistivity Surveys on Various Geological Structures in Illinois. By M. KING HUBBERT. (*Tech. Pub. No.* 463. 8000 *words.*) The results of a summer's field season with a Megger Ground Tester are presented. Positive results were obtained in the location of gravel deposits in glacial drift, anticlines buried under glacial drift, and of faults in Paleozoic strata. Determination of depth to bedrock under glacial drift has not yet been successful.

A Uniform Expression for Resistivity. By SHERWIN F. KELLY. (*Trans.*, vol. 97. 1400 *words.*) A definition of resistivity, and a short discussion of its meaning provide the basis for considering the desirability of adopting a uniform expression for the term. The divergent practices followed by geophysicists to express resistivity are pointed out, and criticisms of some shortcomings are offered. It is shown that the mathematical expression of resistivity, Resistivity, or $\rho = \frac{A}{L} R$, where A is the cross-section of the conductor, L its length, and R its resistance in ohms, leads naturally to the terms ohm-meters, ohm-centimeters, etc. The advantages of adopting the ohm-meter as standard are pointed out, and a plea is made to geophysicists to follow this practice. (The author reports that cogent arguments have since been advanced for employing the expression "meter-ohm," a practice he now advocates and follows.)

Reflection Methods in Seismic Prospecting. By H. M. RUTHERFORD. (*Tech. Pub. No.* 486. 9500 *words.*) The usual method of obtaining depths by the seismic reflection method is given, along with the interpretation of the seismogram on this basis. The relationship between the time-distance curve for refracted waves and that for reflected waves is developed, thus indicating a method of shooting whereby the reflected waves may be identified in terms of refraction "horizons." A formula is developed for the case of several beds. On this basis is shown the validity of using an "average" velocity, along with some indications of the error involved. Some discussion is given relative to the "weathered" layer. The value of using curve-path theory in the interpretation of seismic reflection data is briefly indicated. Actual field data sheets, along with the data and computations, are given. A vertical mechanical seismograph, designed and built by H. G. Taylor, was used in obtaining the data. Reflection records and time-distance graphs are shown to illustrate the various points.

Comparison of Two Methods for Interpretation of Seismic Time-distance Graphs Which Are Smooth Curves. By MAURICE EWING AND L. DON LEET. (*Trans.*, vol. 97. 3700 *words.*) A modification of the Bateman-Herzlotz formula suitable for interpretation of the smooth time-distance curves frequently obtained in seismic prospecting is given. A time-distance curve from the Gulf Coastal plain region is treated by this new method and by the commonly used method of approximating the time-distance curve by a series of straight lines. It appears that the new method enjoys a decided advantage over the older one.

Recent Geothermal Measurements in the Michigan Copper Districts. By JAMES FISHER, L. R. INGERSOLL and HARRY VIVIAN. (*Tech. Pub. No.* 481. 4000 *words.*) Increasing knowledge of heat conductivities and specific heats of the rocks, combined with fresh opportunities to make temperature measurements in deep mines with improved measuring apparatus, led the Michigan College of Mining and Technology to renew work on the problem of the geothermal gradient. It is planned eventually

to make a more complete survey than is here presented. The problem has been attacked both from the point of view of ventilation in deep mines, and with the idea that a relation may exist between natural rock temperatures and mineral content of the formation. The measurements were guided by heat conduction considerations, so as to determine the actual virgin temperatures at the "temperature stations." Holes were drilled a few inches back from the breast, in which were inserted one or more thermometers. Mercury-in-glass thermometers were chosen, after weighing the advantages and disadvantages of various types. Two, and sometimes three thermometers were inserted in these holes, and were read at two-hour intervals until three or four readings had been taken. This proceeding was repeated over a number of days. It was found that reliable readings were obtained without waiting until some days after drilling, and in spite of nearby blasting operations. Measurements in eight special holes, and in a deep diamond-drill hole, all fell nearly on a straight line. The gradient thus obtained is 1° F. in 108.5 ft., or 1° C. in 59.5 m. At 4500 ft. the same rise occurs in 103.1 ft.

A. I. M. E. Technical Publications and Preprints, 1932

All the TECHNICAL PUBLICATIONS and PREPRINTS published in 1932 are available at Institute headquarters. They are also on file in public, university and technical libraries and, when so indicated in the list, may be found in the TRANSACTIONS.

Copies of the TECHNICAL PUBLICATIONS have been mailed to those members of the Institute who are registered in the different classes: A-Metal Mining; B-Milling and Concentration; C-Iron and Steel Division; D-Nonferrous Metallurgy; E-Institute of Metals Division; F-Coal Division; G-Petroleum Division; H-Nonmetallic Minerals; I-Mining Geology; K-Mining Administration; L-Geophysical Prospecting. Other members may obtain copies by writing to the Secretary. Copies of PREPRINTS may also be obtained from the Secretary's office. Additional copies are sold at the rate of 1 cent per page; minimum charge, 25 cents. Nonmembers may purchase the papers at the prices mentioned.

METAL MINING

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE*
484-A.45 F.52	Propeller Type Mine Fan at Moose Shaft, Butte, Montana (13)	A. S. Richardson	New York Feb., 1932	**102**, 86, *93*
487-A.46	Progress in the Improvement of Methods and Equipment at Open-pit Iron Mines on the Lake Superior Iron Ranges (9)	Max H. Barber	New York, Feb., 1932	
Pre.	Sand Filling through Pipes and Boreholes (9)	Lucien Eaton	New York, Feb., 1932	**102**, 33, *41*

MILLING AND CONCENTRATION

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE*
445-B.37 H.19	Soap Flotation of the Nonsulfides (18)	Will H. Coghill J. Bruce Clemmer	New York, Feb., 1932	
461-B.38	Principles of Flotation—An Experimental Study of the Effect of Xanthates on Contact Angles at Mineral Surfaces (48)	I. W. Wark A. B. Cox	New York, Feb., 1932	

IRON AND STEEL DIVISION

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE*
447-C.81	Electrochemical Potentials of Nitrified Steels (31)	Shun-ichi Satoh	New York, Feb., 1932	
450-C.82 E.143	Resistance of Iron-aluminum Alloys to Oxidation at High Temperatures (6)	N. A. Ziegler	New York, Feb., 1932	**100**, 267, *270*
451-C.83	Pressure Welding of Low-carbon Steels with Theoretical Considerations on the Mechanism of Such Welding (42)	C. R. Austin W. S. Jeffries	New York, Feb., 1932	
453-C.84	Inclusions—Their Effect, Solubility and Control in Cast Steel (24)	C. E. Sims G. A. Lillieqvist	New York, Feb., 1932	**100**, 154, *176* *

* Volume numbers are in boldface; page numbers of papers, Roman; and page numbers of discussions, italics. Where no number is given, paper has not appeared in TRANSACTIONS. Discussions are printed following the respective papers. Volume numbers are given as listed on page 3 and in the paragraph at the beginning of the index, page 315.

The number in parentheses following each title indicates the number of pages in the TECHNICAL PUBLICATION or PREPRINT.

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
464-C.85	The Prevention of Intergranular Corrosion in Corrosion-resistant Chromium-nickel Steels (25)	P. Payson	New York, Feb., 1932	**100**, 306, *329*
466-C.86 E.146	Determination of Oxygen, Nitrogen and Hydrogen in Steel (22)	J. G. Thompson	New York, Feb., 1932	
467-C.87	The Equilibrium Diagram of Iron-manganese-carbon Alloys of Commercial Purity (24)	E. C. Bain E. S. Davenport W. S. N. Waring	New York, Feb., 1932	**100**, 228, *250*
468-C.88	Effect of Vanadium in High-speed Steel (9)	A. B. Kinzel C. O. Burgess	New York, Feb., 1932	**100**, 257, *263*
469-C.89	Some Effects of Temperature and Iron Oxide in the Manufacture of Basic Open-hearth Steel (12)	W. J. Reagan	New York, Feb., 1932	**100**, 141, *149*
470-C.90 E.147	Influence of Gases on Metals and Influence of Melting in Vacuo (8)	Wilhelm Rohn	New York, Feb., 1932	
480-C.91	Sintering Economics (9)	Perry G. Harrison	New York, Feb., 1932	**100**, 57, *63*
488-C.92	Paper Withdrawn			
492-C.93	Effect of Small Percentages of Chromium on the Quality of Cast Iron (13)	C. O. Burgess	Buffalo, Oct., 1932	
Pre.	Critical Studies of a Modified Ledebur Method for Determination of Oxygen in Steel (28)	B. M. Larsen T. E. Brower	New York, Feb., 1932	**100**, 196, *224*
Pre.	Effect of Heat Treatment on Corrosion Resistance of Stainless Iron (18)	Clarence G. Merritt	New York, Feb., 1932	**100**, 272, *290*
Pre.	A Quantitative Method for the Estimation of Intercrystalline Corrosion in Austenitic Stainless Steels (9)	J. J. B. Rutherford Robert H. Aborn	New York, Feb., 1932	**100**, 293, *301*
Pre.	Tensile Properties of Rail Steels at Elevated Temperatures (25)	G. Willard Quick	New York, Feb., 1932	
Pre.	The Resistance to Impact of Rail Steels at Elevated Temperatures (10)	G. Willard Quick	New York, Feb., 1932	
Pre.	The Degassing of Metals (28)	F. J. Norton A. L. Marshall	New York, Feb., 1932	
Pre.	The Intermediate Phases of the Iron-tungsten System (15)	W. P. Sykes Kent R. Van Horn	Buffalo, Oct., 1932	
Pre.	Analyses of Inclusions in High-carbon Tool Steels (8)	Haakon Styri	Buffalo, Oct., 1932	

Nonferrous Metallurgy

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
449-D.33	The System PbO-Sb_2O_3 and Its Relation to Lead Softening (12)	C. G. Maier W. B. Hincke	New York, Feb., 1932	**102**, 97, *107*
455-D.34	Sintering Zinc Ore at Rosita, Mexico (8)	H. R. MacMichael	New York, Feb., 1932	
456-D.35	Reverberatory Smelting of Raw Concentrates at the International Smelter, Miami, Arizona (11)	P. D. I. Honeyman	New York, Feb., 1932	
457-D.36	A Comparison of the Use of Various Fuels in Copper-refining Furnaces (18)	E. S. Bardwell	New York, Feb., 1932	
458-D.37	The Messina Stationary Basic Copper Converter (11)	R. G. Knickerbocker	New York, Feb., 1932	
459-D.38	Development of the Leaching Operations of the Union Miniere du Haut Katanga (37)	A. E. Wheeler H. Y. Eagle	New York, Feb., 1932	
471-D.39	Development of Gun-feed Reverberatory Furnaces at Garfield Plant of American Smelting & Refining Co. (7)	R. A. Wagstaff	New York, Feb., 1932	

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
Pre.	The New Lead Refinery and Its Operation at the Bunker Hill Smelter (11)	A. F. Beasley J. B. Schuettenhelm J. W. Johnson	New York, Feb., 1932	

Institute of Metals Division

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
443-E.141	An X-ray Study of the Nature of Solid Solutions (14)	Robert T. Phelps Wheeler P. Davey	New York, Feb., 1932	**99**, 234, *245*
448-E.142	Structure of Cold-drawn Tubing (14)	John T. Norton R. E. Hiller	New York, Feb., 1932	**99**, 190, *201*
450-E.143 C.82	Resistance of Iron-aluminum Alloys to Oxidation at High Temperatures (6)	N. A. Ziegler	New York, Feb., 1932	**100**, 267, *270*
454-E.144	Machinability of Free-cutting Brass Rod (10)	Alan Morris	New York, Feb., 1932	**99**, 323, *331*
465-E.145	Copper-beryllium "Bronzes" (14)	J. Kent Smith	New York, Feb., 1932	**99**, 65, *76*
466-E.146 C.86	Determination of Oxygen, Nitrogen and Hydrogen in Steel (22)	J. G. Thompson	New York, Feb., 1932	
470-E.147 C.90	Influence of Gases on Metals and Influence of Melting in Vacuo (8)	Wilhelm Rohn	New York, Feb., 1932	
472-E.148	Equilibrium Relations in Aluminum-copper-magnesium and Aluminum-copper-magnesium Silicide Alloys of High Purity (13)	E. H. Dix, Jr. G. F. Sager B. Sager	New York, Feb , 1932	**99**, 119, *130*
473-E.149	Equilibrium Relations in Aluminum-cobalt Alloys of High Purity (10)	W. L. Fink H. R. Freche	New York, Feb., 1932	**99**, 141, *148*
474-E.150	Equilibrium Relations in Aluminum-zinc Alloys of High Purity (11)	W. L. Fink Kent R. Van Horn	New York, Feb., 1932	**99**, 132, *140*
478-E.151	A Review of Work on Gases in Copper (28)	O. W. Ellis	New York, Feb., 1932	
485-E.152	Discussion on Some Important Factors Controlling the Crystal Macrostructure of Copper Wire Bars (15)	Discussion of Tech. Pub. 429	Boston, Sept., 1931	
491-E.153	Directional Properties in Cold-rolled and Annealed Commercial Bronze (12)	Arthur Phillips Carl H. Samans	Buffalo, Oct., 1932	
Pre.	The Solubility of Gases in Metals (16)	V. H. Gottschalk R. S. Dean	New York, Feb., 1932	
Pre.	Some Metallurgical Characteristics of Induction Furnaces as Determined by the Absorption of Oxygen by Molten Nickel (15)	F. R. Hensel J. A. Scott	New York, Feb., 1932	
Pre.	Variations in Microstructure Inherent in Processes of Manufacturing Extruded and Forged Brass (9)	Ogden B. Malin	New York, Feb., 1932	**99**, 165, *173*
Pre.	Effect of Small Percentages of Certain Metals upon the Compressibility of Lead at an Elevated Temperature (14)	Lyall Zickrick	New York, Feb., 1932	**99**, 345
Pre.	Effect of Temperature upon the Charpy Impact Strength of Die-casting Alloys (4)	Bert E. Sandell	New York, Feb., 1932	**99**, 359
Pre.	Surface Effects on Assay Beads Caused by Metals of the Platinum Group (17)	J. L. Byers	New York, Feb., 1932	
Pre.	The Degassing of Metals (28)	J. Norton A. L. Marshall	New York Feb., 1932	

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
Pre.	The Role of the Platinum Metals in Dental Alloys (44)	E. M. Wise Walter S. Crowell J. T. Eash	New York, Feb., 1932	**99**, 363, *407*
Pre.	Studies upon the Widmanstätten Structure, III.—The Aluminum-rich Alloys of Aluminum with Copper, and of Aluminum with Magnesium and Silicon (26)	Robert F. Mehl Charles S. Barrett Frederick N. Rhines	New York, Feb., 1932	**99**, 203, *229*
Pre.	On the Theory of Formation of Segregate Structures in Alloys (8)	C. H. Mathewson D. W. Smith	New York, Feb., 1932	**99**, 264, *271*
Pre.	The Magneto-optic Method of Analysis with Particular Reference to the Detection of Elements 85 (Alabamine) and 87 (Virginium) and the Heavy Isotope of Hydrogen (15)	F. Allison	New York, Feb., 1932	
Pre.	Copper Embrittlement, II (11)	L. L. Wyman	Buffalo, Oct., 1932	
Pre.	Some Effects of Internal Stress on Properties of Drawn Brass Tubes (20)	D. K. Crampton	Buffalo, Oct., 1932	
Pre.	A Study of Segregate Structures in Copper-tin and Silver-zinc Alloys (14)	D. W. Smith	Buffalo, Oct., 1932	
Pre.	Properties of Copper Deoxidized with Calcium (11)	Lyall Zickrick	Buffalo, Oct., 1932	
Pre.	The Copper-rich Alloys of the Copper-nickel-tin System (29)	John T. Eash Clair Upthegrove	Buffalo, Oct., 1932	
Pre.	Ears on Cupronickel Cups (7)	W. H. Bassett J. C. Bradley	Buffalo, Oct., 1932	

Coal Division

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
482-F.51	Some Physical Properties of Pennsylvania Anthracite and Related Materials (19)	J. Leland Myer	New York, Feb., 1933	
484-F.52 A.45	Propeller Type Mine Fan at Moose Shaft, Butte, Montana (13)	A. S. Richardson	New York, Feb., 1932	**102**, 86, *93*
489-F.53	Southern High-volatile Coals for Gas and Metallurgical Uses (26)	H. N. Eavenson	Hazleton, Oct., 1932	
Pre.	Moisture Determination for Coal Classification (19)	Edgar Stansfield K. C. Gilbart	New York, Feb., 1932	**101**, 125, *143*
Pre.	Determination of the Alkali-soluble Ulmins in Coal (5)	Edgar Stansfield K. C. Gilbart	New York, Feb., 1932	**101**, 165, *169*
Pre.	Physical and Chemical Properties of Coal in Relation to Classification (26)	H. F. Yancey K. A. Johnson	New York, Feb., 1932	**101**, 171, *197*
Pre.	Condition of Water in Coals of Various Ranks (14)	A. W. Gauger	New York, Feb., 1932	**101**, 148, *161*
Pre.	Some Physical Characteristics of West Virginia Coals (14)	C. E. Lawall C. T. Holland	New York, Feb., 1932	**101**, 100, *114*
Pre.	Application of Ash Corrections to Analyses of Various Coals (12)	A. C. Fieldner W. A. Selvig F. H. Gibson	New York, Feb., 1932	**101**, 224, *235*
Pre.	Properties of Coal Which Affect Its Use in the Ceramic Industry (6)	W. E. Rice	New York, Feb., 1932	**101**, 247, *252*
Pre.	Use Classification of Coal in the Portland Cement Industry (4)	H. P. Reid	New York, Feb., 1932	**101**, 256, *259*

Petroleum Division

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
462-G.38 L.32	Electrical Coring: A Method of Determining Bottom-hole Data	C. and M. Schlumberger E. G. Leonardon	New York, Feb., 1932	

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
490-G.39	Selection and Use of Screened Pipe (21)	Clifford S. Wilson	Ponca City, Sept., 1932	
Pre.	Flow of Air and Gas through Porous Media (20)	Joseph Chalmers D. B. Taliaferro, Jr. E. L. Rawlins	New York, Feb., 1932	**98**, 375, *394*
Pre.	A Method for Computing Pressure Drop in the Pipe of Flowing Oil Wells (35)	K. B. Nowels	New York, Feb., 1932	**98**, 401, *435*
Pre.	Flow of Drilling Mud (11)	H. N. Herrick	New York, Feb., 1932	**98**, 476, *487*
Pre.	Some Experiments on the Behavior of Natural Gas in an Oil-sand Reservoir (6)	Ionel I. Gardescu	New York, Feb., 1932	**98**, 467, *472*
Pre.	Domestic Petroleum Production, 1931 (77)		New York, Feb., 1932	**98**, 127–220
Pre.	Foreign Petroleum Production, 1931 (30)		New York, Feb., 1932	**98**, 221–276
Pre.	Economics of Proration (8)	Joseph E. Pogue	New York, Feb., 1932	**98**, 69, *76*
Pre.	Proration in Texas in 1931 (3)	David Donoghue	New York, Feb., 1932	**98**, 81
Pre.	Economics of Domestic Marketing (16)	Sidney A. Swensrud	New York, Feb., 1932	**98**, 84
Pre.	Some Influences of Foreign Demand on the Domestic Oil Situation (9)	E. B. Swanson	New York Feb., 1932	**98**, 109
Pre.	Developments in Refinery Engineering during 1931 (5)	H. W. Camp	New York, Feb., 1932	**98**, 495

Nonmetallic Minerals

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
444-H.18	Quarry Waste in the Indiana Limestone District (10)	J. B. Newsom	New York, Feb., 1932	**102**, 108, *116*
445-H.19 B.37	Soap Flotation of the Nonsulfides (18)	Will H. Coghill J. Bruce Clemmer	New York, Feb., 1932	
460-H.20	Mining and Treatment of the Sillimanite Group of Minerals and Their Use in Ceramic Products (23)	F. H. Riddle	New York, Feb., 1932	**102**, 131, *151*
475-H.21	Geology of Some Kaolins of Western Europe (22)	E. R. Lilley	New York, Feb., 1932	**102**, 155
Pre.	The Mineral Wool Industry in Indiana (10)	W. N. Logan	New York, Feb., 1932	
Pre.	Magnetic Beneficiation of Nonmetallics (7)	Samuel Gibson Frantz G. W. Jarman, Jr.	New York, Feb., 1932	**102**, 122, *129*
Pre.	Uniform Cost Accounting in the Crushed Stone Industry (9)	William E. Hilliard	New York, Feb., 1932	
Pre.	Results of Wire Saw Tests (5)	J. B. Newsom	New York, Feb., 1932	**102**, 117, *121*

Mining Geology

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
442-I.38	Occurrence of Lead-zinc Ores in Dolomitic Limestones in Northern Mexico (31)	M. W. Hayward W. H. Triplett	Joplin, Sept., 1931	
446-I.39	Ore Deposits of the Tri-State District (Missouri-Kansas-Oklahoma) (46)	George M. Fowler Joseph P. Lyden	Joplin, Sept., 1931	**102**, 206, *239*
452-I.40	Prospecting for Gold in the Shield Areas of Canada, Siberia, Southern Rhodesia and Western Australia (33)	W. H. Emmons	New York, Feb., 1932	**102**, 175
479-I.41	Geology and Economics of Tin Mining in Cornwall, England (17)	E. R. Lilley	New York, Feb., 1932	

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Pre.	Nickel Resources, Production and Utilization (13)	E. S. Moore	New York, Feb., 1932	**102**, 252
Pre.	Geology of the Molybdenite Deposits at Climax, Colorado, and of Other Deposits Producing Molybdenite (9)	John W. Vanderwilt	New York, Feb., 1932	
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